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EIGHTH SESSION.

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ARTHUR C. CLAUDET, A.R.S.M.

(*Member of Council*).

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INSTITUTION OF MINING AND METALLURGY.

EIGHTH SESSION, 1898–99.

FIRST ORDINARY MEETING, 19th October, 1898.

Mr. WALTER McDERMOTT (*President*), in the Chair.

The minutes of the last meeting having been read and confirmed,

The President said that he thought the members could congratulate themselves on the success which had attended the last session. With regard to their *Transactions*, he understood that Mr. Claudet wished to make a few remarks on the subject. The Council had found a great desire to obtain copies of their publications; applications had been received from all parts of the world, and the interest manifested had led to an increasing number of applications for membership.

Mr. A. C. Claudet said that with regard to the papers for the present session, the Council experienced considerable difficulty in getting a sufficient number of metallurgical contributions. Many papers dealing with mining had been received, and, although that was a very important subject, it was highly desirable that metallurgical papers should be sent in. He wished to remind members that the Council were always glad to receive short notes, and, as a matter of fact, it had been found that the discussions on such contributions were often more prolonged and interesting than when a long paper had been read. What was wanted was short notes written by members engaged in practical work, giving working costs, &c., and such contributions could not but be of value, more especially to their students.

"Notes on the Payable Conglomerate Beds of the Witwatersrand, and the Methods adopted for their Extraction."

By A. F. Crosse, M.Inst.M.M.

The chief conglomerate beds of the Witwatersrand district extend east and west of the town of Johannesburg for some 20 miles each way, and probably much further, but beyond the limits just given nothing definite is known, owing to the tilted sandstones containing these beds being covered by rocks of a later date.

These newer rocks do not conform to the older sandstones, and at each end, and where they cover the older stratas, they differ in character.

At the western end these rocks consist of sandstone and dolomite, while at the other end coal measures are met with; in fact it is from this part of the Witwatersrand that the chief coal supply for the gold mines is drawn.

As before stated, the payable conglomerates cover a known distance of some 40 miles. The strike of the outcrop is generally east and west with a dip south. This, however, is true of only about 30 miles of the strike, as beyond this extent the outcrop of the beds curves round to the south before disappearing under the newer rocks.

The above conglomerate beds are locally known as the Main Reef "series." To the north and south of these others exist, but, with the exception of the Reitfontein Bed North, and the Kimberley series, and the Black Reef South, little is known beyond their being of low grade.

The Main Reef series, the one under consideration, is remarkable for uniformity of yield, assay value, strike, and widths of beds, when contiguous mines are compared with one another, but when compared with mines situated miles apart, a considerable

difference is sometimes noticeable, nevertheless, the general resemblance of their chief characteristics is most singular.

The Main Reef series consists of seven different beds of conglomerate, with varying distances between each, and the order in which they occur is approximately as follows, counting from north to south:—

1st Bed of Conglomerate.—The North Reef is a small body of from 8 in. to about 10 in. in width, and its position is generally about 45 ft. to the north of the Main Reef, or North Main Reef Leader, where this exists.

2nd Bed of Conglomerate.—The North Main Reef Leader does not always exist, but where it does it is generally rich. Its position is generally from 2 to 9 ft. north of the Main Reef.

3rd Bed of Conglomerate.—The Main Reef, which is the largest bed of the series, is sometimes of great width, in which case it is generally split, and interspersed with a number of sandstone horses. When the reef narrows down to a few feet in many cases it is found to pay. Its position is generally from 1 to 4 ft. north of the Main South Reef Leader.

4th Bed of Conglomerate.—The South Main Reef Leader is the second reef of chief importance belonging to this series, and next to the South Reef is the richest. Its position is generally from 20 to 30 ft. north of the Middle Reef.

5th Bed of Conglomerate.—The Middle Reef. This reef is not only small in width, but usually carries only a trace of gold. Its position ranges from 25 to 600 ft. north of the Bastard Reef.

6th Bed of Conglomerate.—The Bastard Reef. This reef at times is well defined and carries a little gold. In fact, on more than one occasion it has been mistaken for the rich South Reef. Its position ranges from a few inches to about 9 ft. north of the South Reef.

7th Bed of Conglomerate.—The South Reef, the richest of the Main Reef series, is situated about 60 ft. north of the last reef of the Main Reef series, and this latter rarely carries more than a trace of gold.

Dislocations.—With the exception of the two important dislocations, situated at about 12 miles east and west of Johannesburg, there occur no more of importance until the newer rocks are approached.

In the underground works of the mines overhand, underhand, and breast stoping are carried on sometimes in the same mine, the system used in any part depending on different circumstances, viz., the class of native labour, and the dip of the reefs, and

development. In the early days of the industry, fully 90 per cent. of the natives had to be taught the use of the hammer and drill, and it was found that in stopes which connected with a lower level better results were obtained by underhand stoping than in stoping in any other way. Breast stoping results more from the difficulty of getting the natives to understand the advantage of drilling a hole parallel to the fractured face of the rock below as in overhand stoping. The difficulty the miners in charge experience with natives in this direction will be better understood when it is pointed out that 99 natives out of 100 are unable to construct anything with a square form or with parallel sides, and therefore, although overhand stoping is striven for, an overhand stope in proper order is the exception, and not the rule. In all cases, however, the beds and reefs when narrow are removed with gangue from the stopes in one operation, and it is to this practice, which may admit of improvement, to which attention is drawn.

In removing narrow "beds" or reefs of a few inches in width together with gangue in one operation, it is found that the narrower the reef the greater the quantity of gangue accompanies the pay rock to the mill, even with the most up-to-date washing and sorting appliances, and this condition of affairs not only adds to the cost of milling the pay rock, but adds also to the usual losses of gold left in tailings and slimes, by reason of these being of a greater quantity.

The quantity of gangue going to the mill under the conditions just mentioned would depend more on the width of the reef mined than on the width at which the stope is carried, and perhaps the percentage of waste rocks sorted out, as both the sorting appliance and the narrower width at which a stope can be worked have their limits of efficiency. The miner in a stope requires at least 2 ft. in cross section in which to work, and sorting appliances have to deal with a large amount of broken gangue, which with narrow reefs would be smaller in size than the broken reef, as with these narrow reefs the holes for the blasting charge are put into the gangue. In some mines on the Witwatersrand the South Reef is only 3 in. in width, and where this is the case it is removed with gangue in one operation, and forms a combined width of 2 ft. in cross section. It is obvious that even with 40 per cent. sorted out, a very heavy percentage would still accompany the reef to the mill, and necessitate an assay value of 3 oz. per ton for a 3-in. reef being reduced to 11·41 dwt. per ton.

After considering many of the questions involved in the above practice, it appears possible that departures from the ordinary

system might be introduced by which better monetary results could be obtained, although a higher expenditure per ton would be entailed.

In the case of mining certain narrow reefs which came under the author's observation, the reefs averaged 4 in. in width, with an assaying 47 dwt. per ton. On surface, 40 per cent. of the waste rock was sorted out, and the balance, 60 per cent., and reef, were sent to the mill.

From the width of the reefs mentioned, it will be seen that for every foot excavated in the stopes in the direction of the strike, 576 cub. in. and 2,880 cub. in. were won, or after deducting the 40 per cent. of casing sorted out, 1,728 cub. in. of casing and 576 cub. in. of productive rock were sent to the mill, and therefore the assay value of 47 dwt. of the reef was reduced to 11 dwt. 18 gr.

From practice it is found that ordinary sorting out of the waste rock carries away the gold, but with careful sorting, the loss under this head can generally be limited to 1 dwt. per ton, but this allowance has to be made, and, therefore, 11 dwt. 18 gr. must still be reduced by an amount equivalent to 40 per cent. of 5 dwt. off the original assay value of 47 dwt., which reduces 11 dwt. 18 gr. to 11 dwt. 6 gr.

As regards the sorted-out waste rock carrying away 1 dwt. per ton, it is not certain that the contention is a correct one, but this allowance is made at most of the mines, and even this amount does not account for the whole of the unaccountable losses, and it seems probable that the allowance is not an extravagant one, as it is likely that the crevices and fractures formed in waste rock by blasting become receptacles for freed gold, which would present considerable difficulties to washing and sorting.

The foregoing explains the losses at the mine, and in the items taken from the cost sheet which follows, appear two more constant losses, which occur at the mill and cyanide works.

The following figures do not include the expenditure under the heads of main shaft sinking, main cross-cutting, claim licences, town secretary's office, and directors' fees, but it includes ordinary development, which costs 40*s*. per foot for driving and 30*s*. per foot for raising, a rise being put up at every 200 ft., on account of the broken nature of the ground at that time.

		s.	d.
Mining	per ton	8	6
Pumping and hauling	„	1	6
Development	„	5	7·2
Milling	„	3	6
Cyaniding 70 per cent. of 1 ton ore	„	2	5
Tramming and crushing	„	1	7
Maintenance	„	2	0
Mine office, general charges, and staff	„	2	10
	£1	7	11·2
Loss in slimes, 4 dwt. gold per ton and 30 per cent. of 1 ton		4	2·2
Loss of gold in tailings, 1 dwt. per ton and 70 per cent. of 1 ton		2	5·4
	£1	14	6·8

Assay value of rock milled, 11 dwt. 6 gr. at 3*s.* 6*d.* per dwt., is £1 19*s.* 4·5*d.*, leaving a difference of 4*s.* 9·7*d.* to pay for shaft sinking, &c., as above mentioned.

It is necessary to take an assumed case for mining the ore clean for the purpose of demonstrating by figures the possibility of making some improvement on the present methods practised in mining narrow reefs.

The width of the reef, the assay value, and the losses in slimes and tailings, are the same per ton as in the first case submitted, but on account of the wider stope and the lesser amount of casing sent with the productive ore to the mill, the cost of mining and development is greater.

The chief differences consist in mining the reef in two operations, *i.e.*, removing 2 ft. of gangue first, and leaving the reef with 2 in. of casing to be removed afterwards, and the greater cost of mining and development. In this case it will be seen that 24 in. of casing has been removed from the stope before the reef has been attacked, and that an amount of gangue equivalent to 50 per cent. of the productive ore is sent to the mill. It therefore follows that although the cost of rising and driving is the same per foot, the lesser quantity of casing sent to the mill makes the cost of development higher; in fact, in this case twice as much reef as casing goes to the mill, as against three times as much gangue as productive reef in the first example.

It will also be seen that mining is charged for at 8*s.* 6*d.* per ton on all rock broken and removed for a width of 2 ft. 6 in., whereas in the first case 8*s.* 6*d.* per ton is only charged on a width of 16 in., the amount sent to the mill, although the other 8 in., or 40 per cent. of gangue sorted out, is included in the 8*s.* 6*d.*

The following assumed cost is exclusive of those items excluded in the first case :

		s. d.	£ s. d.
Mining 4 tons of gangue	at	8 6	1 14 0
Mining 1 ton of reef and gangue		..	0 8 6
Pumping and hauling per ton milled		..	0 1 6
Development		..	0 15 8·4
Milling		..	0 3 6
Cyaniding 70 per cent. of 1 ton of ore ..	at	3 6	0 2 5·4
Tramming and crushing		..	0 1 7
Maintenance..		..	0 2 0
Mine office, general charges, and staff		..	0 2 10
Loss in slimes 4 dwt. gold per ton and 30 per cent. of 1 ton		..	0 4 2·2
Loss of gold in tailings, 1 dwt. per ton and 70 per cent. of 1 ton		..	0 2 5·4
			£3 18 8·4

Assay value of rock milled, 31 dwt. 8 gr. at 3*s.* 6*d.* per dwt., is equal to £5 9*s.* 8*d.*, leaving a difference of £1 10*s.* 11·6*d.* to pay for shaft sinking as before mentioned.

From the foregoing it will appear that the figures demonstrate the possible greater economy of mining narrow reefs in two operations, although, in this way, stoping and development would cost much more per ton.

The figures also tend to show that it is possible to obtain by double stoping and a small mill what is obtained by stoping casing and reef together in one operation, and by a mill of more than double the size, and that therefore by double stoping a large amount of expenditure under the head of Capital Account might be saved, without interfering with the amount of gold won and the amount of profit made.

Apart from the non-proved applicableness of double stoping on the Witwatersrand, the author is aware that considerable difficulty would have to be met in the older mines before it could be introduced, and would point out that the sole object of the paper is to obtain information on the subject.

Of the author's personal experience of double stoping he can say little, as the only opportunity offered him occurred at the Langlaagte Estate Gold Mining Company, in 1888, where very good results were obtained by this method, but of which, unfortunately, no proper data can be obtained.

DISCUSSION.

Mr. Crosse said that he wished it to be understood that the paper related to the state of affairs 15 months ago, and that things were very different at the present time.

The President expressed his obligations to Mr. Crosse for his paper. The author had disarmed their criticism by stating that the paper was written some time ago, and his calculations would be entirely altered by the inclusion of the slimes and the larger amount treated by cyanide. He hoped those present who had had experience in South Africa would be able to give figures more up to date on the subject.

Mr. Claudet said that it would be noticed that the proportions in vogue at the time of writing the paper amounted to sending to the mill 3 tons of casing to 1 ton of reef, and in the proposed alteration the proportion would be 2 tons of reef to 1 ton of casing. With such a narrow reef as 4 in. he should have thought that there would be a greater loss than 1 dwt. per ton in the casing. With regard to the author's calculations which set out that in the old series the total cost, including loss of gold, amounted to £1 14*s.* 6·8*d.*, and in the new case to £3 18*s.* 8·4*d.*, it was not clear to him whether Mr. Crosse meant that he was going to mill the same quantity of ore in both instances. The author said:—"The figures also tend to show that it is possible to obtain by double stoping and a small mill what is obtained by stoping, casing, and reef together in one operation," which appeared to indicate that he was not going to mill the same quantity in both cases, but although the figures of cost for milling were identical, he thought the expenses would be much more.

He desired once again to call attention to the manner in which the gold values were calculated. The author stated:—"Assay value of rock milled 11 dwt. 6 gr. at 3*s.* 6*d.* per dwt." And he wanted to know why Mr. Crosse took 3*s.* 6*d.* When assays were the basis of calculation the gold ought to be reckoned at 4*s.* 3*d.* per dwt. No doubt the members had noticed many such instances of late in which calculations were made from assay values, which were always given in fine gold, and stated that their oz. of gold would be at the rate of 3*s.* 6*d.* per dwt., or some other figure. The Transvaal companies avoided, as a rule, giving the fineness of their gold, so that a mine which was turning out, say, 20,000 oz. of gold per month, might not be turning out the same fine gold as a mine yielding 15,000 oz., because in the former case the fineness of the gold might be very much less than in the latter. On 11 dwt. ore he gave the

loss in slimes as 4*s.* 2*d.*; it should be 5*s.* 1*d.*, and the total loss in gold instead of being 6*s.* 7*d.* was 8*s.* per ton. On the 31 dwt. ore it made a very material difference. In the paragraphs immediately following his statements the author said: "Assay value of rock milled 11 dwt. 6 gr. at 3*s.* 6*d.* per dwt. is £1 19*s.* 4·5*d.*" —the actual value was £2 7*s.* 10*d.*—"leaving a difference of 4*s.* 9·7*d.* to pay for shaft sinking"—that figure should be 11*s.* 10*d.* Again: "Assay value of rock milled 31 dwt. 8 gr., at 3*s.* 6*d.* per dwt., is equal to £5 9*s.* 8*d.*"—the real figure should be £6 13*s.* 2*d.* —"leaving a difference of £1 10*s.* 11·6*d.*"—the actual difference was £2 13*s.* 0*d.* Although these corrections were in favour of the author, he drew attention to the matter because it was very important that the extraction should be taken out at the same value as the assay.

Mr. Rickard: It seemed to have escaped Mr. Claudet, that it was only assumed that the rock carried away contained 1 dwt. per ton. That was the basis of the paper; and as that was an assumption, so was the rest of the paper.

With regard to Mr. Claudet's remarks about stoping, he wished to point out that there were several circumstances which had to be taken into account in stripping a vein. It was necessary to know whether, in breaking down the outer rock to uncover the vein, there was a clean wall or let-go. He knew several cases at Johannesberg where a clean wall could not be depended upon. Another case in point was where the contiguous rock itself carried gold, as was the case with the Randfontein mines, where they frequently had a reef of 10 in., and then other 10 in. or so of rock carrying gold, rendering it necessary to break down the wall, rock and reef together.

He would be glad if the author would explain his statement, with regard to the workable conglomerate beds, which were stated to extend for only 40 miles and spoken of as having terminals; as a matter of fact, the conglomerate beds extended for more than 100 miles. There were ideal beds of conglomerate at Heidelberg on the east and Venersdorp on the west, a distance of some 120 miles. How far the banket within these limits may be remunerative is, of course, quite another question.

Mr. E. P. Rathbone said he had generally found with most of the reefs in depth, that there was seldom sufficient selvage to be able to break to any particular point. It frequently happened that one had to blast back very much further, and take out a much greater quantity of the embracing or "country" rock, in order to get some point where a good natural "wall" could be found. Very

often the rocks had to be broken right back to get a good parting so as not to endanger the lives of those below.

Mr. Claudet thought that the following extract from the Report of the Roodepoort and United Main Reef Gold Mining Company, would be interesting, showing that that Company hoped to be able to sort out 35 per cent. of waste rock.

> "It is the intention of your Board, as soon as possible, to institute a policy of close and systematic sorting in place of the existing arrangement. The narrowness of the South Reef makes it necessary to mine waste in the proportion of three of waste to one of reef—*i.e.*, the broken reef represents only 25 per cent. of the total mined product. The facilities now existing on the mine provide for about 8 per cent. sorting only. If this were increased to from 30 to 35 per cent., the yield per ton, which amounted for the year to 42*s*. 8·48*d*., would show a material increase."

The President said that if the conditions were such that it was possible to break up the waste rock to a sharp line, no mine manager would mix it up with his rich ore, in order to send it up to the surface to be picked out again and milled. There must always be the difference between mine sorting and hand picking, and the fact that the latter was making increasing progress in the Transvaal, and all the new mills were fitted with a special apparatus for reducing the amount of waste crushed to a minimum, showed that the necessity for the removal of the waste was recognized, and if it could be overcome in a simple manner in the mine itself by a sharp division of waste and rock it would be done. In the first place, by leaving it underground there would be the saving of the cost of sending it to the surface and also of the cost of crushing. The supposition that the waste rock could be broken to within 2 in. of the payable vein was, in his opinion, unfounded, otherwise it would have been carried into effect.

He agreed with Mr. Claudet in his remarks as to the uncertainty of the values, and considered that all calculations in a paper pretending to scientific accuracy should be made on the basis of fine gold, so as to make comparisons possible with all parts of the world. The production of low grade gold by the cyanide process had caused a little confusion in the minds of many people, and he thought that the Institution, as a scientific body, should do its best to insist on all calculations being on the basis of fine gold. It was being recognised in the Transvaal that this was a step in the right direction, and the tendency was to reduce values to that basis.

Mr. A. G. Charleton asked whether the assay value of the ore milled, 11 dwt. 6 gr., given by the author was fine gold or bullion.

Mr. Crosse said that the object of his paper was to see whether it was not possible to obtain better results by doing the mining in two operations. If a comparison was being made with a different system altogether, it might be perhaps important that the value should be taken into account, but as the same reef was being dealt with he did not think it mattered very much. It was true that, if the assay value was taken, a fraction over 4*s.* per dwt. should be reckoned instead of 3*s.* 6*d.*, but in most cases on the Rand they were glad to allow a small margin or an allowance for the unascertained losses of gold.

Under the head of unascertained losses, some, but not the amounts, could be traced to the following:—

Gold dissolved and lost, when the plates are dressed with cyanide and other solutions.

Gold in very fine particles carried away by water.

Gold carried away on sickened mercury from the plates and boxes.

Gold, or amalgam, lost in cleaning plates at the usual daily and weekly and monthly clean-up.

In reply to Mr. Rickard's remarks, the reef was split into two, and the hanging wall contained about 3 dwt. The reef was resting on a thin sheet of slate, which he believed could be readily taken away.

In reply to Mr. Charleton's question, the figure of 11 dwt. 6 gr. was fine gold.

Mr. Alfred James said that in regard to Mr. Claudet's remarks, that gentleman appeared to wish all gold to be taken at the London Bank value, but if that were done, insurance, freight, bank and realization charges would have to be taken into account.

The President observed that Mr. Crosse's position seemed to be that for the purpose of comparison it did not matter which was taken.

Mr. Claudet asked whether the two tables of cost were based on milling the same quantity of ore.

Mr. Crosse replied that they were.

Mr. Mactear said that it seemed to him that the figures in the first table, referred to milling 4 tons instead of 1.

Mr. Crosse stated that they were milling 1 ton with 75 per cent. of casing.

Mr. Mactear thought that the milling costs should be increased,

seeing that they were set out as 3*s.* 6*d.* for 1 ton milled, whereas 3 tons of barren rock was milled with it.

Mr. Crosse replied it went in with the mixture.

"Notes on Lode Tin Mining in the Malay Peninsula."

By W. H. DERRICK, Assoc. Inst.M.M.

ALTHOUGH two papers (one by the late Mr. H. M. Becher, and one by Mr. Frank Owen) have already been read before the Institution of Mining and Metallurgy on the subject of *alluvial* tin mining in the Malay Peninsula, a few brief notes on the working of the principal *lode* tin mining company in the Federated Malay States, viz., the Pahang Corporation, Limited, may be of interest, especially to Cornish tin miners, who, although vitally interested in tin mining in the Straits, in consequence of the large output from the alluvial mines there having lowered the price of tin so greatly, still learn nothing from the information given in the interesting papers referred to above regarding the working costs of *lode* mining, and cannot therefore make any comparison between cost of lode mining in the Malay Peninsula and in Cornwall, although in both countries the work is carried on under almost identical conditions, the same treatment being necessary in order to obtain from the stone mined tin oxide in a marketable form.

The Company above mentioned holds in Kuantan, Pahang, one of the Federated Malay States, a concession estimated to contain over 100 square miles. Although tin, both lode and alluvial, has been found over the greater part of the concession, the payable lodes appear to lie in the comparatively small area of about 10 square miles. These lodes run approximately east and west, are from 50 to 2,000 ft. apart, range in thickness from 2 to 10 ft., and produce from 1 to 15 per cent. of tin oxide to the ton.

The lodes underlie both north and south, at angles ranging from 10° to 40° from the vertical; they have generally one well-defined wall, but seldom two. The geology of the district is similar to that of parts of Cornwall, the granite, the basement rock, being much the same and being also overlaid by varieties of clay schist, with the tin lodes running down right through the slate into the granite. A few isolated hills of calc-spar remain, although at one time this rock

must have covered the slate, as the latter is everywhere seen intersected with veins of spar.

The principal metals found in the lodes are tin, copper, iron, and arsenical pyrites, blende, and galena. Of these metals tin and copper only have been discovered in workable quantities; the latter usually carries from 25 to 30 oz. of silver to the ton, and appears to be quickly giving place in depth to tin.

The mines now being worked by the Pahang Corporation were, so report says, continuously worked by Malays and Chinese for more than 100 years, the large surface excavations made by them, some of which are 1,000 ft. long, 200 ft. wide, and 150 ft. deep, testifying to this being a fact. The open-cast system was the only one adopted by the old miners, and as timber was seldom made use of to secure loose ground, the sides of their working were sloped or terraced to keep them from falling in. This, and not the thickness of the lodes, accounts in many instances for the great width of the old workings. Under native management (at least within recent years) the mines were not a financial success, owing apparently to want of capital, and, consequently, of proper machinery, as well as the native objection to doing anything in the shape of dead work, their mode of operation being to follow any payable ground from the surface down as far as they were able to, without steam pumps and timber.

Explosives of any description were never made use of, owing to a strange superstition firmly believed in by the Chinese, viz., that the use of explosives frightens away the metal in a mine. In consequence of this, any very hard ground could not be worked. The ore obtained was crushed by wooden stamps shod with iron and worked by small overshot and undershot water wheels. The only dressing appliance used was a Long-tom, the tail losses from which were very heavy. These tails were assayed, and found to contain as much as 7 per cent. of tin oxide. They are principally composed of coarse grains of tin and quartz adhering together, and which only required finer crushing to allow the tin to be readily separated out.

For working the mines, the Pahang Corporation make use of Chinese, Javanese, Malay, and Tamil labour, Europeans being employed for supervision work only. With the exception of ore dressing, all work is let out on contract.

Stoping is paid for by the truck of 16 to 18 cwt., prices delivered at the battery ranging from 1*s.* 9*d.* to 6*s.* per ton, according to the nature of the ground stoped and the distance of the mine from the battery, which in some cases is as much as 2 miles, the mines being all connected with the batteries by tramroads.

Contractors pay for all tools and materials, dynamite, fuze, detonators, candles, &c., and put in timber as required.

If the quality of the ore delivered at the battery drops below a certain percentage (which is fixed at the time of letting the contract), the contractor receives nothing for such ore. This insures the Company against the delivery of waste. Each contractor's parcel of ore is stamped separately, and is constantly sampled and assayed as it passes through the mill.

Driving and sinking are let by the fathom, and prices range from £1 10*s.* to £4 for drives, 7 ft. by 5 ft.; £2 10*s.* to £5 10*s.* for winzes, 6 ft. by 5 ft.; £5 to £30 for shafts, 12 ft. by 4 ft., these dimensions being all clear of timber.

Contractors, as in stoping, pay for all stores and put in timber as required, the timber of course being supplied free by the Company. Drills are sharpened for contractors free. At one time this work was charged for, but it was found men wasted much time by using blunt tools rather than pay to have them re-sharpened.

The distances driven per month, by a gang of six men working in 8-hour shifts, vary from 10 to 40 ft. according to the nature of the ground.

Shaft sinking ranges from 10 to 20 ft. per month in hard slate when 12 men are employed working 6-hour shifts.

The Chinese miner makes good progress in moderate and soft ground, but is usually a poor miner in really hard rock, and on this account he cannot compete with white labour. If time alone was the sole consideration in mining this might be so, but when expense also is taken into consideration it is far otherwise, a fact which is very apparent when one considers that white labour when employed on actual mining work is about ten times as expensive as native labour; in other words, a white man has to drive or sink ten times the distance done by a native in order to compete with him, whereas in any but hard ground his progress would probably not much exceed that of an ordinary skilled Chinese miner.

The average cost of mining, including cost of drives and winzes (but not permanent shafts), timber, hauling and pumping charges and European supervision, is 5*s.* per ton.

The Pahang Corporation have two batteries of 40 and 20 head of stamps respectively. The stamps are of the usual Californian type, weight 850 lb., and driven at a speed of 90 blows a minute, they crush about 2¼ tons per head per day. Vertical, high pressure, non-condensing engines supply the motive power.

At present some 2,500 tons of tin stone are being treated per month for a yield of 85 to 95 tons of tin oxide—an average of about

3½ per cent. per ton. The ore is only crushed fine enough to pass through screens having 80 to 120 round burr holes to the square inch. Both copper and Swedish iron grates are used.

The dressing appliances first introduced were the usual concave and convex buddles from 12 ft. to 25 ft. in diameter, frames and tossing gear as seen on the dressing floors throughout Cornwall; but notwithstanding the cheap labour available for working these, they are gradually being replaced by more modern machinery and with the best of results.

Frue vanners with corrugated belts, size 4 ft. by 12 ft., are found to answer well in treating the ore as it comes direct from the stamps without classification. Working thus on ores carrying a large percentage of pyrites, the tables give heads (containing 25 to 45 per cent. of tin oxide) sufficiently clean to go direct to the calciner without further handling, and with tails ranging from 3 to 5 lb. to the ton. Three tables are employed to treat the output from 10 heads of stamps. The heads from the vanners are much cleaner than those obtained from the buddles, and the introduction of vanners has resulted in a reduction of at least 50 per cent. in the quantity of raw concentrates returned to the calciners, and, consequently, is a corresponding saving in roasting charges.

For calcining, the ordinary reverberatory furnace is used, and, with wood at 10*s.* per cord (128 cub. ft.), cost of roasting, including labour, is 4*s.* per ton of concentrates treated.

Tin-dressing coolies cost 18*s.* per month. Native engine-drivers, carpenters, and blacksmiths £2 to £3 per month. Europeans for supervision work £20 to £30 per month. The total dressing cost, which includes stamping, European supervision, native labour, stores, and roasting charges is 5*s.* 6*d.* per ton of stone crushed.

The standard to which the oxide is dressed always exceeds 70 per cent. of metal, the impurities being oxide of iron, a little silica, and from $\frac{1}{16}$ to $\frac{1}{8}$ per cent. of copper.

The total battery losses range from 5 to 8 lb. to the ton.

Two pulverisers are employed grinding some 250 tons of coarse concentrates per month. One frue vanner is found sufficient for each pulveriser. Plain belt machines, belts 4 ft. by 12 ft., are found to give good results with the fine ore from the pulverisers, returning a very clean head and the tails, not exceeding 6 lb. to the ton, when working on concentrates, running as high as 10 per cent. tin oxide to the ton.

A separate engine is employed for driving the vanners, and a high level tank is used for the water supply for same, thus ensuring a

regular speed and steady flow of water—two very essential points in the successful working of frue vanners.

Jiggers of the 3-compartment type were tried at one of the mills, but did not give satisfactory results, and were consequently discarded.

When it is considered that lode tin mining in the Malay Peninsula is only as yet (under European management) in its infancy, it is but natural to expect that the next 10 years will show a marked increase in the present output of tin from this source. The few details given above show that the mines here can well compete—in the way of working expenses—with the home industry, and with the benefit of past experience future discoveries will doubtless be more rapidly developed and working expenses reduced to such an extent as to enable very low grade ores to be profitably treated; and in bringing about these results the numerous waterfalls and quick-running streams found all over the peninsula will, by providing cheap power, play an important part.

DISCUSSION.

The President thought that the paper was a very interesting one, more especially that portion of it dealing with the advance which had been made in such a distant country in the method of concentration, and which was at the present time also being adopted in Cornwall.

Mr. J. H. Collins considered that Mr. Derrick's paper was very tantalising, inasmuch as it failed to give a good deal of information which one might have expected to be given. The author stated that the lodes ran in an east and west direction, and pass through stratified rocks into the granite, but said nothing about the distribution of the rich parts of the lodes. No information was given as to how much ground had to be broken to get a ton of stuff for the stamps. As to the distribution of the ore in the veins, the author did not say whether the tin, copper, and lead ores occurred in definite "shoots" or "courses"; whether in one mixed mass, or in separate leaders that could be picked out and treated separately. No information was given as to the depth of the existing workings, except that some of the open workings were down to 150 ft. Mr. Derrick also said nothing about the amount of water met with, or the kind of ground—whether it had to be supported with timber.

He would like to know what the author meant by hills of calcspar; he had never seen such hills, and supposed the author was referring to limestone hills containing veins of calcspar.

With regard to tramming, he should like to know the method employed—whether by horse, steam engine, or man.

The cost of mining was put down at 5*s.* per ton; he should like to know whether that sum meant the cost of mining proper or of quarrying. The cost of stoping was put down as 1*s.* 9*d.* to 6*s.* a ton; even the lower figures would be very high for quarrying.

With regard to the stamps, the tailings were said to contain 7 per cent. of tin; he thought that most extraordinary, and could only suppose that small quantities of natural concentrates were found in the stream that washed away the tailings, which would go as high as 7 per cent.

As to the dressing costs, in spite of the Frue vanners and cheap labour, they amounted to 5*s.* 6*d.* a ton, which was extremely high.

He would like to know how the author arrived at his figures of 5 lb. to 8 lb. per ton as battery losses, in order to check the calculation for himself.

It was evident that the jiggers failed because they were worked by some one who did not know how to use them; an ore containing 70 lb. or 80 lb. to the ton, of coarse grains of tin oxide, and which only had to pass through holes $\frac{1}{16}$ in., could be jigged very easily, and with great advantage.

With regard to the final paragraph in the paper, he did not think it was possible at present to usefully compare tin mining in the Malay Peninsula and Cornwall, for the conditions were not sufficiently known. He did not doubt that if there were numerous lodes in the Malay Peninsula, giving an average all over of $3\frac{1}{2}$ per cent. of black tin, the production would increase considerably in the next few years, but at present there was no reason to suppose that such was the case. The author stated that the payable lodes were comprised in an area of 10 sq. miles, but a good deal more would have to be known of the mining geology of the country before comparisons could be made such as the author suggested. At present the proper mining costs seemed to be very high, and although Cornwall has been hard hit by the extremely abundant and rich alluvials, the cheap labour, and enormous water supply in the Malay Peninsula and islands, most Cornishmen felt that, as far as the evidence went at present, they had not much to fear from lode mining competition anywhere.

Mr. Thos. Rickard said that although the paper was, no doubt, a valuable one, it ignored alluvial mining, which was the one thing

home mining had most to fear. In the locality dealt with by the author, there appeared to be plenty of water and any quantity of rich alluvial matter. In a paper read by Mr. Frank Owen some time ago, it was stated that a start was made to hydraulic, and it was desirable for those engaged in home tin mining to know whether that kind of mining was going to succeed; if it did, Cornwall would have to look out.

Mr. Collins should have included in his statement of the advantages enjoyed by the Malay Peninsula over Cornwall the fact that the home industry had to compete with a depreciated currency, which had a good deal to do with the difficulty under which it was labouring. He thought that there was, perhaps, no other country in the world which would allow one of its oldest industries to be pressed out of existence for the want of a little effort to save it. He asked what had become of the gold about which a good deal was said in former papers, but to which no reference was made by the author.

Mr. W. S. Lockhart observed that stamps and pulverisers were mentioned by the author; he would like to know what particular kind of pulverisers were used. With regard to the calcspar hills mentioned by Mr. Derrick, to which description Mr. Collins had taken exception, he had not been in the Malay Peninsula, but he had been in Burma, and the formation of the two countries was very similar. In Upper Burma every one was familiar with these hills of calcspar, often 800 ft. to 1,000 ft. high, and generally containing extensive caves.

The calcspar was the well known white translucent mineral, breaking up readily into rhombohedral crystals, and was not to be confused in any way with mountain limestone.

Mr. A. G. Charleton pointed out that the dressing cost only worked out at 1*s.* 6*d.* a ton, because the 5*s.* 6*d.* referred to by Mr. Collins included roasting charges, which were stated to be 4*s.*

The President agreed that there was a good deal left unsaid in the paper, but reminded the members that it only claimed to be a series of notes, and still left it open for a longer and more detailed paper to be sent in. With regard to the tailings, said to contain 7 per cent. of tin oxide, he took it that it was certain of the residues of dressing which required re-grinding in order to separate the tin. It could hardly be said that the crushing was coarse; the author stated that the ore was crushed to pass through screens having 80 to 120 round burr holes to the square inch; he should call that fine crushing.

He thought that perhaps Mr. Derrick might be asked to sup-

plement his paper, and give the information that had been asked for.

Mr. Claudet remarked that the author stated that his notes were written to supplement other papers already published, and possibly the information desired was to be found in the previous papers.

"Notes upon Mining and Concentration of Copper Ores in the Lake Superior Region."

By P. R. ROBERT, M.Inst.M.M.

THE principal working mines of the Lake Superior copper region are situate on a promontory extending northward into the lake, known as Keweenaw Point, and are so much on one side of the main lines of transcontinental travel, that the district scarcely receives the attention it deserves.

The writer deems that a brief *résumé* of some operations conducted there during the past year may prove of interest to the members.

Almost the entire copper product of the lake region is derived from (1) Conglomerate beds and (2) Amygdaloid beds; the amygdaloids being a species of trap or melaphyre.

The most productive of the former is the Calumet conglomerate of the Calumet and Hecla and Tamarack Mines.

The principal amygdaloid mines are the Quincy, Osceola, and Atlantic, each working a separate bed of amygdaloid.

The ore is mined upon a large scale, each class of cupriferous beds having a thickness of from 8 to 20 ft. as a general rule. The ore is crushed by rockbreakers at the mine and then carried by railway from 4 to 10 miles to mills, which are situated on the shores of the lake. There the ore is stamped by steam stamps, the mortars having steel screens with holes $\frac{3}{16}$th of an inch in diameter; the stamped ore passes through hydraulic separators of the V pattern, and from them to the jigs, the ore passing over the ends of the separators goes to slime settlers, and the settlings are passed over Evan's slime tables.

The stamps weigh about 4,000 lb., have a stroke of about 24 in., and make 90 blows per minute. The ore of from 1 to 3 per cent. copper contents is concentrated to yield from 70 to 80 per cent. of copper. The concentrates are carried to the smelting works and smelted in reverberatory furnaces. Its treatment may be defined as

simple fusion, and the product of the smelting works is shipped for the most part to New York, where it is sold.

The Calumet and Hecla Mine treats the richest ore on the lake, and next comes the Tamarack, working on the same deposit.

The lowest grade of ore worked in the district is at the Atlantic Mine.

The author proposes to deal with the operations at the Atlantic and Tamarack Mines during the year 1897.

The Atlantic Mine.—The work done was :—

Shafts sunk.....	210 ft.
Winzes.....	68 „
Drifting	5,937·5 „
Total	6,215·5 „
Stoping	18,501 cub. fathoms.

Total ground broken in openings and stopes	24,429 cub. fathoms.
„ ore milled (2,000 lb.-tons)	394,296 tons.
„ product of mineral (75·51 per cent.)...	6,766,960 lb.
„ yield of ingot copper	5,109,663 „
Yield of refined copper per fathom	209 „
„ „ „ ton of ore	12·96 „
Gross value per ton of ore (6*s.*).....	$1·46
Working cost „ „ (5*s.* 8·67*d.*).....	$1·18$\frac{9}{10}$

		Cents.
The net cost of mining, selecting, and breaking (including taxes, $9870·06)...	per ton	73·43
Transport to mill (9 miles).....	„	4·54
Milling and concentration	„	23·94
Freight, smelting, marketing, and N.Y....	„	17·03
Construction account	„	10·75
Total expenditure, including construction	„	129·69
The sales of copper averaged	per lb.	11·23

The details of cost of milling and concentration were for 394,296 tons (the mill has six stamps) as follows :—

Labour.....	$28,520·20	=	7·23	cents per ton.
Fuel.....	53,580·00	=	13·58	„ „
Supplies	9,173·32	=	3·13	„ „
Insurance	1,560·00			
Teaming mineral, &c.	1,542·92			
	$94,376·44	=	23·94	„ „

The Atlantic amygdaloid has an average width of 12 ft. and an inclination of 56° from the horizon. The shafts are inclined, being on the dip of the lode, and the workings are entirely in the amygdaloid. The deepest workings are a few feet below the 26th level (about 1,950 ft. vertical depth).

The net receipts of the mine in 1897, after paying for construction and exploration $42,394·20, and for land purchased $7,000, were $57,077·43.

The amygdaloid rock has been considered easier to stamp than the harder conglomerate, but this is not the case. Although it is softer it lacks the brittleness of the conglomerate; the wear of shoes and dies however is much less.

The Tamarack Mine.—The problem here is an entirely different one. The conglomerate does not crop out on the property, but it has a dip of 37½° from the horizon. The mine is worked entirely through vertical shafts and cross-cuts. The lode was first reached some years ago at a vertical depth of 2,560 ft. The depths of the shafts are now as follows:—

No. 1 shaft	.....	3,240 ft.
„ 2 „	.....	3,775 „
„ 3 „	.....	4,526 „
„ 4 „	.....	4,450 „
„ 5 „	.....	2,015 „

In 1897 there were hoisted 726,665 tons of rock. Of that amount 54,947 were from cross-cuts, shafts, and winzes, 671,718 tons were broken from the conglomerate, and 60,179 tons were rejected, 611,539 tons were milled, and 25,908 tons were milled for another mine. The yield of mineral from 611,539 tons of ore milled was 29,583,380 lb. ($68\frac{35}{100}$ per cent.).

Yield of ingot copper was	.....	20,222,559 lb.
„ „ per ton of ore	.....	33·06 „
Value per ton of ore (estimated at 11·23 cents)	.....	$3·71 (or 15*s.* 3*d.*)

The cylinders of the steam stamps mentioned in this paper have a diameter of 20 inches. The stamps weigh about 4,000 lb., and have a stroke of 24 inches, making 90 blows per minute. The stamp stems are 8 in. in diameter.

Of late years the greatest improvement made in the steam stamps has been the substitution of solid foundations for the mortar beds, in place of the spring timber beds formerly used.

The work done during the past year was:—

Shafts sunk	1,280·5 ft.	
Winzes	827·0 „	
Drifting	6,471·3 „	on conglomerate.
Ditto	1,420·9 „	on Osceola lode.
Cross-cuts	3,445·0 „	
Total	13,444·7 „	

In Shaft No. 4 the conglomerate was cut at 4,185 ft., and at about 4,740 ft. this shaft should cut the Osceola amygdaloid, which has the same dip and lies about 730 ft. to the eastward of the Calumet conglomerate. This bed is worked with good results by the Osceola Company, and extends through the Calumet and Tamarack properties. The mill has seven heads of steam stamps, and last year the running time was equivalent to one head running 1,914½ days, and each head crushed nearly 333 tons (2,000 lb.) per day (a very excellent result), making the total crushing 637,447 tons.

In the company's report the gross expenses were given as $1,835,809·86, made up as follows:—

		Tons.
Mining, transport, &c.....	$1,104,359·94	(726,665)
Milling and concentration	161,525·79	(611,539)
Smelting, marketing, freight, &c. ...	365,001·40	
Construction account	204,922·73	
Total	$1,835,809·86	

Costs per ton:—

	Tons.	Cents.	Tons.	Cents.
Mining, transport, &c.....	(726,665)	151·9	(611,539)	180·5
Milling and concentration ...	(611,539)	26·4	„	26·4
Smelting, marketing, &c. ...	„	59·6	„	59·6
Construction	„	33·5	„	33·5
		271·4		300·0
		$2·714		$3·000

The net income for the year was $431,530·89.

From the foregoing it is seen that the actual cost of producing a ton of milling ore was $3·00, less 33·5 cents paid out for construction = $2·66½ (or 10*s.* 11·4*d.*).

The most noticeable fact in relation to the operation of steam stamps is the small cost of repairs. This has always been so, as the moving parts are few in number and friction is reduced to a minimum. Another item in their favour is the small floor space

required for the large amount of work performed. Another strong point is the fact that in crushing the ore a smaller proportion of slimes is made than with other stamps.

As regards the cost of mining in depth on a flat lode with vertical shafts, we have an example in the Tamarack. The old bugbear of largely increased charges *on account of great depth* is exploded in these days.

DISCUSSION.

Mr. Mactear called attention to the very small amount of copper in the ore to start with, the yield of refined copper per ton of ore being only 12 lb.—very little more than ½ per cent. of the original ore. The yield at the Tamarack Mine was set out as 33 lb. (practically 1½ per cent.), and he thought that both the figures were extremely low to represent payable ore.

Mr. E. P. Rathbone asked the President whether compound engines were used, or only single cylinder. He understood that the stamps were fully 5,000 lb. dead weight, instead of 4,000 lb., as mentioned by the author.

The President believed that single cylinder engines were used. With regard to the stamps, it was probable that the solid foundation which had been substituted for the mortar beds had lessened the necessity for such heavy stamps.

Mr. Rickard considered that $\frac{3}{16}$ in. was a size which could more advantageously be treated by a roll crusher, and it had the advantage that the machinery was obtained with less expense, and less slimes were made while only the same power was required. It occurred to him also to ask whether it would not be better, when dealing with a continuously rich reef with a dip of nearly 40°, and sometimes less, to put down the main working shafts on the dip.

He disagreed with the author's statement that the "old bugbear of largely increased charges on account of great depth, was exploded," and thought Mr. Robert would have great difficulty in proving his assertion. One might decide the matter in such an offhand manner if there was no water to deal with, and the ore occurred in large masses, whose value could be calculated out with something like precision. Even on the Rand, where there was relative uniformity in the quality of the ore, it was by no means immaterial; in many cases they were experiencing difficulty, which would increase in the future. Even when leaving the question of interest and amortisa-

tion, he contended that the questions of water and ventilation would tell very seriously against attempts to lay out the works of mines where the richness and mass of ore could not be calculated beforehand.

Mr. A. G. Charleton said that, if he was correctly informed, the reason the shafts were sunk vertically was because the Company did not own the outcrop of the lode, and shafts had, therefore, to be put down to intersect "the reef" on the dip.

The President observed that they began to mine at 4,000 ft.

Mr. Charleton remarked that they had to put down the shafts in the first instance in order to strike the upper portion of the reef within their property. He did not agree with the author's concluding remarks, and thought that great depth did materially increase costs. He could not corroborate Mr. Rickard's remarks as to the relative duty of stamps and rolls, and thought that that gentleman would find the horse power expended in crushing the rock would be considerably less with stamps than rolls. Another objection to the use of rolls was the fact that native copper ore had to be dealt with, which, owing to its malleability, could not be crushed with rolls at all.

The President agreed with Mr. Charleton that it was absolutely out of the question to treat the rock with crushing rolls. The rock was exceedingly hard, and bound together by strings of metallic copper, which would simply smash any ordinary rolls, or would keep being returned without any possibility of final reduction. The rock itself was largely composed of jaspar of an exceedingly hard character, and all sorts of stamps and Chilian mills had been used, and many years of experience had proved that steam stamps were most economical.

Mr. Charleton had practically answered the question of the vertical shaft. The Tamarack Mine was working on the deep level of the Calumet and Hecla, and had to sink 4,000 ft. before it could begin to mine at all. He thought that what the author meant by his final paragraph was that it had been proved that one could start mining at a depth of 4,000 ft. on a proposition which was not exceedingly rich in itself, as was shown by the grade of the ore mentioned, and that with modern appliances for economical hoisting and rock drills, which greatly assisted the question of ventilation in deep working, it was possible to work and get people to find the money, which in former years would have been considered out of the question. Only a few years ago people were in doubt whether it was possible to mine at a depth of 4,000 ft. on the Rand. The Tamarack began at 4,000 ft., and the bore holes which had been put down proved

that there was no great increase in the temperature. No large volume of water had been met with, and the consequence was that works were being now undertaken for a vertical depth of 5,000 ft. and an incline depth at the bottom of that of an additional 3,000 ft. The question of water could of course only be dismissed when one was dealing with a mine that had none; if it were met with in any quantity it would of course add to the cost.

SECOND ORDINARY MEETING, 16th November, 1898.

Mr. WALTER McDERMOTT (*President*) in the Chair.

The minutes of the previous meeting having been read and confirmed,

The President stated that as they had two papers dealing with similar subjects, it might be advisable that both should be read before discussing them, the subsequent discussion to include both papers.

"Notes on Dry and Wet Crushing with Cyanide Treatment, in New Zealand."

By JOHN McCONNELL, M.Inst.MM.

Dry Crushing.

Drying.—The ore before crushing is thoroughly dried, which is generally done in kilns, although rotary dryers are much better for the purpose. The kilns are made by excavating a circular pit on the side of a hill, tapering towards the bottom, with a discharge door into a tunnel, through which the quartz is discharged when dry and trucked into the battery.

These kilns are charged with an alternate layer of firewood and ore. The bottom layer of firewood is then lighted, and in time lights the other layers and thoroughly dries the ore between. As the ore is dried it is gradually withdrawn from the bottom and more ore and timber filled in at the top, so that the process of drying, once commenced, is continuous.

The quantity of firewood used is 1 ton to from 3 to 4 tons of ore. The cost of drying ranges from 2*s.* to 2*s.* 6*d.* per ton.

This method of drying for some classes of ore does well enough as a "make-shift," but, as compared with other methods, is crude, expensive, and unsatisfactory. With ores containing even a small

percentage of sulphides difficulties are created that interfere with the after-treatment.

The ore being charged into the kilns in alternate layers with the firewood is subjected to a most uneven heat. Portions of it are severely roasted, and other portions little more than dried, so that a most irregularly dried product is obtained. The sulphides are irregularly decomposed, and varying complex reactions take place, with the formation of free sulphuric acid, sulphates and oxides of ron, that are decidedly objectionable in the treatment.

Another serious difficulty is the formation of charcoal in the kilns, which is discharged and crushed up with the ore, and precipitates during treatment a portion of the gold taken up by the cyanide solution, and so lowers the extraction.

Again, the heat in kiln drying cannot be controlled, but depends entirely on the dampness of the ore and the quantity and nature of the timber charged with it.

What is required for drying ore, to be treated afterwards with cyanide, is a rotary dryer so arranged that the heat can be controlled and regulated to the requirements of the ore.

Crushing.—The ore, generally speaking, is exceptionally hard, and crushing by stamps is almost universally adopted. The ore after drying is put through a stonebreaker or, preferably, a stonebreaker and Gates crusher, so as to reduce it to as fine a grade as possible before being fed into the stamps. The stamps on the average weigh from 900 lb. to 1,000 lb. each, and are driven at the rate of 94 blows per minute with a 6½-in. drop. The screen generally used is 30-mesh steel wire screen (900 holes to the square inch), and the quantity of ore crushed equals 1 to 1·25 tons per stamp-head per day of 24 hours.

The grade of ore obtained crushing through a 30-mesh screen is as follows :—

Passed 30 mesh retained on 40	.....	1·20
„ 40 „ „ 60	.....	9·00
„ 60 „ „ 90	.....	12·56
„ 90 mesh	.....	77·24
		100·00

It will thus be seen that, although a 30-mesh screen was used, over 77 per cent. of the ore product was crushed fine enough to pass through a 90-mesh (8,100 holes to the square inch). The discharge power of a dry stamp is entirely due to the concussion of the blow, which throws the crushed ore against the screen, but only portions

of which pass through, the rest falling back into the box. This, in addition to the limited area of screen for each stamp, accounts for the ore product obtained being so much finer than the screen used. In ores that do not require such fine crushing for a high extraction, a coarser screen can be used, which will largely increase the output. Fine crushing, however, for the bulk of the New Zealand ores is absolutely necessary, as the larger proportion of the gold in them exists in such an extremely fine state of division that to expose every particle to the action of the cyanide it is necessary to reduce them to almost an impalpable powder. But there is a limit with each kind of ore, which can only be arrived at by experiment, as to how far it is profitable to reduce the output of the stamps by crushing fine for the purpose of increasing the extraction.

The crushed ore is conveyed from the stamps by means of a pipe conveyor or otherwise, and charged evenly—so as to insure an equal percolation—into the vats to a depth of from 1·5 to 2·5 ft. The solutions are carefully run on from below, and the washings are drawn off by means of a vacuum. The solutions and washings, after treatment, are passed through zinc turnings and the gold recovered in the usual way. Each charge takes from 7 to 10 days to treat, although with some refractory ores a much longer time than this is necessary. The tailings after treatment are sluiced out and passed over amalgamated plates to catch any coarse gold that the ore may contain, and then, if necessary, over blanket strakes to catch any untreated coarser pyrites, that are dealt with afterwards in pans or Berdans, or smelted if rich enough. In ores containing acid salts lime is added as a neutralising agent to the stamp box, and gets thoroughly mixed with the ore while being crushed. To leach first with an alkali such finely crushed ore interferes with filtration, and, through the settling of the ore, frequently prevents an even percolation and equal exposure to the cyanide solution.

The objections put forward against dry crushing are—

1. The extra cost of drying the ore.
2. The dust difficulty in crushing and in charging the vats.
3. The reduced output of the stamps.

The dust difficulty is a serious one, as the floating dust from the stamp boxes and from the vats while charging penetrates into every corner of the plant, and where it settles on the bearings causes incredible wear and tear. It also seriously affects the men, as by their continually breathing the finely divided quartz in the form of floating dust their lungs get affected. The dust escaping from the stamp boxes can be overcome by having a suction fan connected to

the outside casing of the box by means of a pipe which draws it off as it escapes and deposits it in a settling chamber. The dust from the vats while charging is more difficult to deal with, but could be lessened by having a jet of steam or a very fine spray of water playing over the vat while being charged, which would prevent the dust rising to a great extent. When the whole of the floating dust from an ordinary dry crushing battery is allowed to escape, a considerable loss takes place, and more especially, as is the case with some ores, such as the Waihi and Waihi-Silverton, where the escaping dust assays about twice the value of the crushed ore saved. This shows the extremely fine state in which the gold exists in the ores of these two mines, and the absolute necessity of fine crushing.

With regard to the smaller output of the stamps in crushing dry as compared with crushing wet, this is a disadvantage. When the same screens are used in both cases the output is largely in favour of wet crushing if a plentiful supply of water is used, but the fineness to which the ore is reduced is in favour of dry crushing. A greater, but coarser, output is obtained by crushing wet caused by the assistance of the flow of water in the discharge. If it is necessary that the ore should be reduced wet to the same degree of fineness as that obtained dry, either a finer screen must be used or the flow of water reduced. Even then, however, although not to the same extent, the output is in favour of wet crushing.

To wet crush ores containing a large proportion of fine gold, with water there is always a great danger of float loss.

Wet Crushing with Cyanide Solution.

In wet crushing with a cyanide solution as little solution as possible is used, from 1·5 to 2 tons to the ton of ore. The ore pulp on leaving the stamp-head box is passed over amalgamated plates to catch any coarse gold, and is conveyed by means of a launder into revolving distributors which discharge the ore evenly into the vats.

As the ore pulp is being charged into each vat the ore settles, and the surplus solution, which contains more or less slimes, is allowed to overflow, and passes through two or more settling vats for the purpose of settling the slimes, and then through the zinc precipitators, and is pumped back again to the storage vats, where it is made up to the required strength and used over again. When sufficient ore has been charged into the vat the ore is allowed to settle, which is hastened by the addition of a little lime. The settling of the ore takes from half an hour to an hour and a half. The clear solution is then run off from the top to within a couple of inches from the

top of the ore, and the vacuum applied to draw off the remaining solution through the ore. The washings are applied and drawn off in the usual way. It is advantageous to add a little lime to the stamper box while the ore is being crushed; this helps the settling of the ore and slimes in the vat, and allows the overflowing solution into the settling vats to go off comparatively clear, and after passing through the first settling vat is usually as clear as desired.

The object of crushing wet with a cyanide solution is that the ore while being crushed is subjected to a brisk agitation in the stamper box and also while passing along the launder through the distributer into the vat. This greatly assists and hastens the solution of the fine gold, and cleans any coarser gold the ore may contain, making it more amenable to amalgamation on the plates.

With sulphide ores, in which the gold is difficult to dissolve, and which require either a lengthened exposure to the solution or a brisk agitation, the following continuous method of treatment could be applied successfully. Crush the ore with a cyanide solution and pass over plates as described; then run the ore pulp into a series of agitators, so arranged that the ore pulp would overflow continuously through the agitators from the one to the other until each particle of ore on passing through would have the necessary agitation for its treatment. Convey the overflowing pulp from the last agitator into a vat, and filter and wash as already described.

Frequently in giving heavy sulphide ores a lengthened percolation, there is considerable difficulty, as the ore after the first few days settles down hard in the vat, which is apt to make filtration and washing slow and imperfect.

The quantity of cyanide solution used for crushing wet is necessarily much greater than in dry crushing, but the actual consumption of cyanide per ton of ore treated in a properly constructed and well managed plant after the first solutions have been made up and the plant thoroughly saturated is as near as possible equal to the consumption in the treatment of dry crushed ore.

The quantity of ore crushed per stamp through a 30-mesh screen, using from 1·5 to 2 tons of solution to the ton of ore, averages 1·25 to 1·5 tons per stamp per day of 24 hours.

The following are a series of experiments made by crushing with stamps through different mesh screens with different quantities of solution.

	30-mesh.	40-mesh.	60-mesh.	80-mesh.	100-mesh.
Left on 40	1·0				
„ 60	8·0	3·5			
„ 80	11·1	10·5	17·2		
„ 90	5·7	3·5	3·2	2·4	
„ 100	5·0	9·5	11·6	3·8	
Passed through 100	69·2	73·0	68·0	93·8	100·0
	100·0	100·0	100·0	100·0	100·0
	Ton.	Ton.	Ton.	Ton.	Ton.
Crushed per stamp	1·48	1·51	1·06	1·15	0·38
Solution used per ton of ore crushed	1·60	2·00	2·80	4·80	8·0

The following experiments also will show the variation of the grade obtained in crushing with different quantities of solution through the same mesh :—

	With 4 tons solution of ore to the ton.	With 2 tons solution to the ton of ore.
Retained on 60-mesh	13·5	3
„ 90 „	20·5	21
Passed on 90 „	66·0	76
	100·0	100

There is a much greater wear and tear of screens in wet crushing than in dry, and crushing with a 60-mesh screen (3,600 holes to the square inch) is impracticable, no matter how carefully the stamps are fed, owing to the bursting of the screens.

The following experiments were made with concentrates from Woodstock ore crushed wet, and after passing over the plates and crushed fine enough to pass through a 90-mesh sieve, show the advantages of agitation treatment over percolation for a speedy extraction.

	Solution.		Time in hours.	Concentrates per ton.		Tailings per ton.		Extraction.	
	Before.	After.		Gold.	Silver.	Gold.	Silver.	Gold.	Silver.
	KCy.	KCy.							
	p. c.	p. c.		oz.	oz.	oz.	oz.	p. c.	p. c.
Percolation	1·2	1·0	70	0·5	13·9	0·15	10·65	70	23·4
"	"	"	"	"	"	0·15	10·65		
"	0·6	0·45	"	"	"	0·18	12·62	64	12·5
"	"	"	"	"	"	0·18	11·72		
Agitation..	1·2	1·1	16	"	"	0·05	2·4	90	83·8
" ..	"	"	"	"	"	0·05	2·1		
" ..	0·6	0·49	"	"	"	0·08	4·37	82	66·5
" ..	"	"	"	"	"	0·10	4·95		

It will be seen from the above that the extraction by agitation is much higher than by percolation, although by percolation treatment the concentrates had over four times the length of exposure to the cyanide solution. The strength of solution required for a high extraction should also be noted.

The following experiments were made with the same class of refractory ore from which the above concentrates were taken, but from which all the pyrites had been eliminated. It is evident that this class of ore (Woodstock ore) freed from the pyrites is just as difficult to treat as the concentrates, and requires as strong a solution for a high extraction.

The ore was crushed wet by stamps, through a 30-mesh screen.

Hours agitated.	Solution.		Ore per ton.		Tailings per ton.		Extraction.	
	Before.	After.	Gold.	Silver.	Gold.	Silver.	Gold.	Silver.
	KCy.	KCy.						
	p. c.	p. c.	oz.	oz.	oz.	oz.	p. c.	p. c.
18	1·2	1·1	0·25	3·0	0·03	0·67	88·0	77·7
18	0·1	Trace	"	"	0·15	2·4	40·0	20·0
18	1·2	1·09	0·23	4·47	0·03	1·12	87·0	75·0
18	0·1	0·02	"	"	0·1	2·95	56·6	44·0
18	1·2	1·1	"	6·12	0·03	1·37	87·0	77·7
18	0·6	0·52	"	"	0·05	2·55	78·3	58·0
18	0·1	0·013	"	"	0·1	5·25	56·6	14·3

The following are the results of a few experiments made with the

same class of ore, but without concentration. The ore pulp was caught (after being crushed by stamps through a 30-mesh screen, and after passing over the amalgamated plates), and agitated with a portion of the solution it was crushed with. The experiments were made on 2-ton charges.

Hours agitated.	Solution.		Ore per ton.		Tailings per ton.		Extraction.	
	Before.	After.	Gold.	Silver	Gold.	Silver.	Gold.	Silver.
	KCy.	KCy.						
	p. c.	p. c.	oz.	oz.	oz.	oz.	p. c.	p. c.
	1·2	1·13	0·55	7·8	0·21	5·84		
1	..	..	..	..	0·18	5·57		
2	..	..	..	..	0·11	4·42		
3	..	..	..	..	0·11	4·92		
4	..	..	..	..	0·1	4·68		
5	..	..	..	..	0·08	4·22		
6	..	..	..	..	0·05	3·65		
7	..	1·1	..	..	0·05	3·55	91	54·5
	p. c.	p. c.	oz.	oz.	oz.	oz.	p. c.	p. c.
	1·2	1·11	0·42	7·98	0·24	6·39		
1	..	..	..	..	lost			
2	..	..	..	..	0·18	6·05		
3	..	..	..	..	0·16	5·89		
4	..	..	..	..	0·14	5·54		
5	..	..	..	..	0·13	5·2		
6	..	..	..	..	0·11	4·92		
7	..	..	..	..	0·1	4·7		
8	..	1·11	..	..	0·08	3·72	81	53·4
	p. c.	..	oz.	oz.	oz.	oz.	p. c.	p. c.
	1·2	..	0·7	9·4	0·15	7·45		
1	..	..	..	..	0·15	6·5		
2	..	..	..	..	0·13	7·17		
3	..	..	..	..	0·13	7·72		
4	..	..	..	..	0·1	7·2		
5	..	..	..	..	0·[illegible]	7·5		
6	..	..	..	..	0·1	6·95		
7	..	..	..	..	0·08	7·07		
8	..	..	..	..	0·08	6·82		
11	..	1·11	..	..	0·08	6·47	88·6	31·2
	p. c.	p. c.	oz.	oz.	oz.	oz.	p. c.	p. c.
1	1·2	..	0·48	7·75	0·25	6·13		
1	..	..	..	..	lost			
2	..	..	..	..	0·16	4·89		
3	..	..	..	..	0·13	5·2		
4	..	..	..	..	0·1	4·95		
5	..	..	..	..	0·09	4·36		
6	..	..	..	..	0·08	4·32		
7	..	1·08	..	..	0·08	3·27	83·4	57·8

Hours agitated.	Solution.		Ore per ton.		Tailings per ton.		Extraction.	
	Before.	After.	Gold.	Silver.	Gold.	Silver.	Gold.	Silver.
	KCy.	KCy.						
	p. c.	p. c.	oz.	oz.	oz.	oz.	p. c.	p. c.
	1·2	..	0·53	8·43	0·32	7·68		
1	..	..	..	..	0·2	6·53		
2	..	..	..	..	0·11	5·52		
3	..	..	..	..	0·16	5·72		
4	..	..	..	..	0·13	5·6		
5	..	..	..	..	0·09	4·96		
6	..	..	..	..	0·09	5·44		
15	..	1·1	..	..	0·05	2·2	90·6	73·9

The above ore contained not more than 0·3 per cent. of pyrites, a small percentage of manganese, and a trace of copper. By percolation treatment with an exposure to the solution of five or six days, not more than 65 per cent. of the value was extracted. This shows that with ores such as this an agitation treatment is necessary for a high extraction.

With different classes of ores different methods of treatment must be adopted, so as to obtain a maximum extraction at a minimum cost. Sometimes it is profitable in dealing with certain classes of ore to increase the output at the cost of a lower extraction, and again in other ores the very opposite is the case. In some cases wet crushing should be adopted, and in others dry crushing will be found to be most profitable. It is therefore necessary when a treatment plant has to be erected, instead of blindly following the treatment adopted at a neighbouring plant, which is too frequently done, that the ore to be dealt with be thoroughly investigated, and its proper treatment determined on beyond doubt before the plant is started. When a plant once erected has to be altered to suit the requirements of the ore, the working of it is never satisfactory or profitable.

"Notes on Dry and Wet Crushing."

By Frank Merricks, M.Inst.M.M.

The subject of dry *versus* wet crushing has recently been so prominently before the minds of mining engineers, that the author has been prompted to bring it before the members of this Institution, with a hope that they will contribute their views on the matter, and thus perhaps be the means of saving many a manager much worry and expense in deciding whether he should adopt dry or wet crushing for the treatment of the particular class of ore he has to deal with.

The author is fully alive to the fact that a process or plant ought to be adapted to the ore for its proper treatment; but in this respect mistakes have occurred, and are often duplicated in the same district, by one Company following the mistake of another.

The physical conditions obtaining in any particular district have also to be seriously considered in deciding on any ordinary or special treatment of ore; and perhaps, in certain cases, the process best adapted to the ore has to give way to one less suitable, as regards treatment, owing to existing conditions.

A great impetus to dry crushing was given by the introduction of rolls and ball mills, which, in many of the best designs, are ill adapted, or totally unsuited, to wet crushing. The merits claimed for this class of mills are:—

They cost less, are easier and more quickly erected, produce a pulp of a more even consistency, and require less power, than stamps.

These merits are probably well deserved and applicable in some districts and with certain ores, but, in the long run, it is a very open question if they are more economical than stamps, even as dry crushers.

Dry stamp crushing and cyaniding has been carried on on an extensive scale at several of the mines in the Upper Thames district of the Hauraki Peninsula, New Zealand, and so far as the extraction

of the bullion is concerned, with fairly satisfactory results. The percentage of extraction of the bullion by the ordinary amalgamation process was so small, that to use this method was hopeless ruination.

A short outline of the processes tried and results obtained, and the eventual adoption of dry crushing and cyaniding at the principal mine, the Waihi, may be of interest.

The first two tons of ore from this mine were wet crushed (about 1880), and amalgamated on copper plates, the bullion saved being 35 per cent. of the original value of the ore (£4 4*s.*). During seven or eight years after this, 18,000 tons of ore were treated by wet crushing and amalgamation for an average return of 13*s.* 6*d.* per ton. The author does not know what percentage of the total bullion contents this represents.

During some four years later, 1890–94, a series of experiments were made. The ore was treated by pan amalgamation, and both dry and wet crushing were resorted to, with the result that in the former case 70 per cent. of the gold and 45 per cent. of the silver, while in the latter, 60 per cent. of the gold and 25 per cent. of the silver, were saved. It is reported that the duty of the stamps in each case was practically the same, but by wet crushing the calcined ore, the output was increased 30 per cent., while the extraction still remained 10 per cent. in favour of dry crushing. In 1894 pan amalgamation gave way to cyanide treatment, the ore being dry crushed, with the result that an average extraction of 90 per cent. of the gold and 50 per cent. of the silver was effected.

These results, corroborating those obtained at the Crown Mine in the adjoining Karangahake district, were considered so satisfactory, that the companies in the surrounding districts, where the ores are of a somewhat similar nature, adopted this process, without for a moment considering the advisability of trying raw wet crushing and cyaniding, or even attempting any modification whatever.

A move in the right direction was made about 18 months ago, at the Crown Mine, where wet-crushing experiments were carried out with cyanide solution passing through the mortars.

The method of handling the slimes and the results obtained are not yet made public, but they may be considered satisfactory, since the whole plant has been converted to wet crushing.

The question of converting mills from dry into wet crushing is often attended with difficulties, *i.e.*, in mills where it is intended to run the pulp direct from the mortar to the vats, as, in some cases, the top of the vats is higher than the discharge of the mortar.

At the Waihi Mine the stamps weigh 900 lb., and with a 6-inch

drop, and 90 to 94 drops per minute crush, through a 40-mesh screen, about 1¼ tons per stamp per 24 hours.

These figures and results are those generally obtaining in the district.

The low results obtained by ordinary amalgamation are accounted for by the fact that the gold is in an extremely fine state of division in the ore, and that the proportion of silver to gold in the bullion is excessively large, the fineness being about 640. Very rarely can any gold be detected by the naked eye, or even with the aid of a magnifying glass, or in panning.

The differences in the above results, both as regards dry and wet crushing, and the extraction returns of such, by treating the ore by the cyanide method, call for more serious comment. It is generally admitted that drying the ore previous to crushing renders it more amenable to the cyanide treatment, probably owing to the fact that, although the pulp is reduced to such a fine state of division in crushing, it is of a more porous nature, which is mainly brought about by the dehydration of the ore. This may partly account for the difference of 10 per cent. in the extraction results, although the author is inclined to the belief that it may also be due to the floating away of some of the sulphide of silver, and of the fine gold, and also to the want of a proper method of sampling.

The increase in the output by wet crushing of only 30 per cent. over dry crushing is rather remarkable; probably this increase was looked upon as the maximum duty of the stamp which, combined with the fact that the increase of 10 per cent. in the extraction in favour of dry crushing, was considered so satisfactory that wet crushing entirely gave way to dry crushing.

Since the matter of wet crushing has been so seriously taken up by the Crown Company, most of the other companies are following its example, and although, in some cases, the results are said to have been indifferent, there is every indication that during the next year most of the ore will, and ought to, be wet crushed.

The author is of the opinion that the improvement effected at the Crown Mine should be completed by adopting wet crushing, amalgamation, and cyaniding in its entirety. By this means the output would be greatly increased, say to 3 tons, for a less cost than at present dry crushing 1¼ tons, and supposing the percentage of extraction is a little less by wet crushing, it would be more than counterbalanced by the increased output.

The bulk of the Upper Thames ores present no greater obstacles to wet crushing than did those of the Rand. Dry crushing has been repeatedly tried at different mines in the latter district, but

has been abandoned in favour of wet crushing; in fact, were it not for the high stamp duty, it is very certain that the greater part of these ores could not be profitably worked.

In drying the ore in the crudely constructed kilns, excessive heat often becomes localised, and occasionally the bullion gets melted, which, in the form of small pellets, is not dissolved by the weak cyanide solution.

At the present time a great divergence of opinion exists as to whether dry or wet crushing is the better and more economical for treating the Kalgoorlie ores, West Australia, and in the author's opinion this problem is not yet solved.

In the early days of this field, when the surface or oxidised ores were treated, wet crushing and ordinary battery amalgamation were principally resorted to for extracting the gold, but owing to the fine state of division in which a great percentage of the gold existed in these ores, together with a short supply of water for battery purposes, only a poor extraction was effected, thus leaving the tailings very rich, many of which have been, and are now being, treated by cyanide. As depth was attained and tellurides, carrying a high percentage of gold, were met with, combined with the paramount water question, serious obstacles were presented in devising the best methods for the most efficient and proper means of treating these somewhat rare and complex ores. The most prevailing idea, at present, appears to be that they ought to be dry crushed, roasted or *vice versâ*, and then subjected to cyaniding.

Great difficulties, however, are met with in adopting this course, the first being the coarse gold, while in roasting, the gold in the rich telluride forms into pellets, in both of which cases the gold will be left in the tailings after cyaniding, and to extract it, a further treatment will have to be employed.

In roasting, there is the risk of loss of the precious metals by volatilisation, and in cyaniding, any tellurium left in the ore will be dissolved by the cyanide and precipitated by the zinc; thereby causing a further loss of gold by volatilisation in roasting the precipitate.

Taking into consideration the general characteristics of these ores, notably their friable nature, their free gold (often coarse) contents, which extend below water level, and their association with rich tellurides, it would appear that the above treatment is somewhat involved, necessitating much extra handling of the ore and very careful manipulation in connection with the roasting, and consequently rendering expenses somewhat high.

The method which suggests itself as being the best and most

suitable for the handling and treating of these ores is, first to hand-pick the rich tellurides and to bag such for smelting. In regard to crushing, to break down the ore to a small size by means of crushers, and then to stamp wet, and to guard against sliming crush as coarse as possible consistent with economical working and general conditions obtaining and to collect as much gold as possible by ordinary amalgamation; then to concentrate the tellurides and bag for smelting; the tailings, if carrying sufficient gold, being treated by cyanide, and the slimes collected by, and treated in, filter presses.

DISCUSSION.

The President said that the Council were very pleased to have the papers which had been read, as they dealt with a subject on which there was a diversity of opinion. With regard to Mr. McConnell's paper, it was not an advocacy of either the wet or the dry process, and admitted the necessity of modifications according to the nature of the ore. The chief importance of the paper consisted in the information given regarding the working of an ore which was difficult to treat and had hitherto been crushed dry, and which was being experimented upon in connection with wet crushing.

Mr. Merricks's paper was a suggestive one, and slightly enlarged the field of discussion by bringing in Western Australia, and directly throwing down the gauntlet on the method of treatment by suggesting a process which was opposed to what was at present being adopted in the best mines.

Mr. Alfred James considered that it was a great advantage for the Institution to have a communication on the subject of wet and dry crushing in New Zealand from the man who first introduced the cyanide process into that colony, and who, having resided in the particular district for a number of years, was thoroughly acquainted with the matter under discussion.

With regard to the output by wet crushing as contrasted with that obtained by dry crushing, Mr. McConnell stated that the output, using a 30-mesh screen, was practically 1½ tons per head per diem with a solution ratio of 1·6; he thought this was a very small amount of water in comparison with that used elsewhere. In the countries of big outputs the quantity of water was frequently eight times the amount of ore crushed, and he desired to know

whether any experiments had been made with varying proportions of water, and, if so, whether it made any difference in the output.

Mr. McConnell stated in the early part of his paper that the output by wet crushing was practically the same as by dry crushing, and all the figures given by various authorities seemed to bear out that statement; that was altogether contrary to what would have been expected, and it occurred to him that one reason for it was that, owing to the use of cyanide solution in the boxes, the amount of water used was so small as to prevent the free passage of the ore through the screens. That might not be the explanation—it might be due to the inherent toughness of the ore—but it was a most important matter for consideration. Seeing that Mr. Merricks in his paper confidently assumed (without giving any figures in support of his assumption) that the extraction by wet crushing could be raised to 3 tons per head per diem, it seemed to him that Mr. McConnell's paper was very much an answer to the suggestions made in Mr. Merricks's contribution. The latter gentleman assumed that the Crown Company's wet crushing was a success, and that therefore all the batteries in New Zealand, Western Australia, and other places ought to crush wet. No figures were given to show that it was a success, and as far as appeared from the paper they might be still experimenting. According to Mr. McConnell, however, the present attempt appeared to be a failure. In the first place the output was not increased, which was the main reason for the adoption of wet crushing, and in the second place the ore when crushed was not treatable by cyanide in the ordinary way. Mr. McConnell gave as a reason the fact that the wet crushed product was of coarser grade than the dry crushed product through the same screens. If that were so it followed that the wet crushed product of equal grade would be less per head per diem than the dry crushed product of equal grade, and such was the case at Waihi, in the experiments alluded to by Mr. Merricks. It was stated that in the early days, when both wet and dry crushing were used, a similar amount was produced by the wet and dry processes, but it was found that they could use coarser screens with dry crushing, and the output was thus greater than the wet when a 30 or 40-mesh screen was used as against that of 60-mesh, which was required for the wet method. It seemed, therefore, that the wet crushing process in New Zealand had not been a success either by increasing the output or improving the extraction, nor were the costs reduced.

The proportion of water used to the ton of ore appeared to be limited because of the large extent of dam and other accommodation

necessary. If six times as much water as at present were used (as in South Africa) there would be six times as much cyanide liable to decomposition when not treating the ore, and consequently there would be a greatly increased consumption cost for cyanide; there would also be six times as much gold constantly circulating in the solution, which itself would be six times as dilute in passing through the zinc boxes. It was therefore to their interest to limit the amount of solution as much as possible, even if it limited the amount of ore crushed. This all told against the success of the process from a crushing point of view, which was the main reason for its adoption.

In the early part of his paper Mr. McConnell remarked that "fine crushing for the bulk of New Zealand ores is absolutely necessary," and that was the principal obstacle to the success of the wet process. In carrying out some experiments to ultimate extraction point—95 per cent. extraction—he had found it necessary to pass all particles through a 90-mesh screen in order to get a complete extraction. That would be practically impossible with wet crushing, because with screens so fine the output would be very much less in comparison with that by dry crushing.

It seemed too that with this coarse wet crushed product to deal with the author was driven to devise a new process for its treatment. The percolation process was no longer efficacious with particles which were more or less coarse, and therefore the agitation process had to be adopted. He considered that Mr. McConnell's remarks on the matter were borne out by other experiments on the treatment of this particular ore. He did not think that these results necessarily showed that wet crushing in the Ohinemuri district could not be made successful, but rather that the present method of treating the ore was not a success, and recent investigations threw additional light on the results which were being obtained. Some people who had been exploring the river beds found that just where the tailings came out of the Crown Mines the river bed assayed 28*s*. per ton; he would not say that that necessarily had any connection with the results of the method of treatment at the Crown Mines, but it was rather a curious coincidence. Formerly when the Crown Mines were using the dry process their extraction was from 90 to 93 per cent., and their output about 24 cwt. per head per diem. When wet crushing was adopted their output was no more, and he understood that their extraction was considerably less—about 85 per cent. If the process was to be made a success it was absolutely essential that the ore should be treated on skilled lines, and not all run together—coarse, sands, and slimes—into one vat, as they appeared to be

doing at present. The coarser particles, which would always be in evidence in wet crushing, would be separated, just as on the Rand, and receive special treatment: the sands would receive ordinary treatment, and the slimes further special treatment unless the agitation in the mortar boxes had been sufficient. The pulp should be classified, and each portion given a chance of successful treatment. At present only a limited time was given for the lixiviation of the whole mass, so that there was always a portion of the particles which had not received sufficient time, and therefore they were sluiced away into the Ohinemuri River while still containing a large portion of their gold.

Mr. Merricks in the early portion of his paper said:—"These results, corroborating those obtained at the Crown Mine in the adjoining Karangahake district, were considered so satisfactory that the companies in the surrounding district, where the ores are of a somewhat similar nature, adopted this process without for a moment considering the advisability of trying raw wet crushing and cyaniding, or even attempting any modification whatever."

According to the history of the process in New Zealand that statement was scarcely correct, and he should like to have Mr. Merricks's explanation. What happened was that almost every mine at work in the district—the Crown, Te Aroha, Waihi, Waihi-Silverton and Komata and Woodstock properties—originally used wet crushing, but one after another they made experiments and as a result adopted dry crushing. The statement regarding dry crushing on the Rand was too sweeping, as Mr. Merricks could see by looking up the monthly returns of the Luipaard Vlei Company.

Mr. Merricks stated that the output was practically the same whether wet or dry crushing was used, but that by calcining the ore the output with wet crushing was increased by 30 per cent.; but that method was open to the very objections Mr. Merricks was endeavouring to avoid, viz., the extra cost of calcining and the melting of the bullion into the form of insoluble pellets.

The point he desired to emphasise was, that Mr. Merricks quietly assumed, without giving any figures or data, that because with an exceptional ore, which was perhaps the one ore in the world best adapted to the process (the Rand "Banket" ore), 5 or 6 tons per head per diem could be crushed and treated by amalgamation and cyanide; it could also be done on an exceptionally difficult ore, such as was found in New Zealand. Mr. Merricks said that the bulk of the Upper Thames ores presented no greater obstacles to wet crushing than those of the Rand, but he (the speaker), thought that he had not taken into account the nature of and the mineral carrying

constituents of the ore. In South Africa the ores were very friable and easily crushed, but in the Ohinemuri district they were so tough that mills which easily crushed Rand ores failed entirely to touch them. The main difficulty with the Ohinemuri ores lay in the fact that they contained a very large percentage of sulphide of silver, which could practically not be treated by cyanide; it dissolved with such exceeding slowness, that it had not yet been efficiently treated. There was generally, with frequent exceptions caused by rich patches of silver, a definite ratio between the gold contents and the sulphide of silver, and it appeared to be five or six of silver to one of gold, and he suggested that it was this combination of gold with the practically insoluble sulphide of silver which put out of the question the treatment of the Karangahake ores by such a simple process as that used on the Rand.

With regard to Western Australia, it was news to him that cyanide solutions dissolved tellurium, and deposited it in the zinc boxes, and this action, if present at all, must surely be very minute; in fact he understood that high authorities stated that oxide of tellurium was not soluble in cyanide solutions, and that this characteristic was the basis of the method for depositing selenium from tellurium, so much used by analysts. Mr. Merricks suggested treating Western Australian ores by stamping with light stamps—and he seemed to disregard the fact that the telluride ores slimed very easily, and the result of that treatment would be that there would be much tellurium in the form of very fine slimes mixed with coarser particles of sand, which in practice it was almost impossible to concentrate, even with any close sizing, Mr. Merricks also did not seem to take into account the fact that it had been proved by actual work that it did not pay to smelt telluride ores which carried less than 4 oz. of gold to the ton in comparison with the results which could be obtained on the spot. The smelting charges averaged from £5 to £6 per ton, which put it out of the question, except on very rich ores.

In conclusion, he ventured to suggest that Mr. Merricks had overlooked the recent immense development of coarse dry crushing in Africa, in districts where the ores were too poor to pay for wet crushing. This treatment appeared to be the salvation of some of the Kantoor and Lydenburg mines, and it was being adopted also in Swaziland, Klerksdorp, and even on the Rand itself.

Dr. A. L. Simon said that it appeared to him that the figures given in the tabulated statements in Mr. McConnell's paper were calculated upon the gold which remained in the tailings, and that could not be regarded as a practical extraction from an ore. The gold

remaining in the tailings, and the actual amount of gold recovered, ought to be given. On Rand ores in the laboratory the results obtained very often showed 85 to 90 per cent. extraction, whereas the practical extraction in good cases only worked out to 60 or 70 per cent.

As to the dust question, he considered it was due in a great degree to failures in construction of plant, and he knew a great many dry crushing plants where not a particle of dust was to be seen. As he had said, he could only put the dust nuisance down to the faulty designing of a plant. Several dry crushing plants were working in Western Australia at the present time, giving great satisfaction and a high extraction, particularly the one working at the Hannans Brownhill, which seemed to be a very good one, although it was difficult to obtain any detailed information with regard to it. He hoped shortly to be in a position to lay before the Institution some details with regard to dry crushing in Western Australia. One reason why dry crushing was resorted to in that country was, that the water sometimes contained from 20 to 30 per cent. of solid matter, which naturally rendered any wet process particularly difficult.

With regard to roasting the gold zinc slimes when telluride was present, he understood that it had been abandoned, and instead of converting the zinc into oxide of zinc, and getting it into the slag, a wet process had been selected which gave much more satisfactory results.

Mr. J. H. Collins thought they had a great deal still to learn with regard to the question of dry *versus* wet crushing, and he looked forward with interest to the additional paper which had been promised by Dr. Simon on the subject. From the particulars they had before them, it was not possible to come to any very definite conclusions.

With regard to the drying of the ore for dry crushing, he considered that it ought not to be a serious obstacle, and he suggested the employment of a large, revolving tube of the Oxland calciner type, which he thought would reduce the cost from 2*s.* to 4*d.* or 5*d.* per ton. It seemed to him that one reason why the dry crushing referred to by the author was more effective than wet, was that the ore was not merely dried but roasted, which greatly facilitated the crushing. The question of discharge was also of importance. If there was a proper exhaust arrangement for carrying off the fine particles as fast as they were made, not only would the dust nuisance be got rid of, but the speed of crushing would be greatly increased. In the case of wet crushing, if there was a large

flow of water, it was possible to crush a great deal more stuff in the same time, and with the same number of stamps.

They were told that dry crushing gave a larger proportion of fine material than wet crushing with screens of equal mesh. Similar results were obtained in Colorado in wet stamping by the use of a very high drop of the stamp. He knew a mill where the same kind of highly pyritic ore was treated with the slow-drop stamp and the quick-drop stamp in different parts of the mill; the latter made about 80 or 85 falls per minute, while the former made about 30. The result, roughly speaking, was that the slow-drop stamp gave more gold on the plates, more slimes, and less gold in the concentrates, than the quick-drop stamp. The total result, however, was rather in favour of the slow-drop stamp on that particular ore, notwithstanding a slightly higher milling cost.

Mr. E. P. Rathbone considered that where there was any possibility of getting water, dry crushing should be avoided, as being most undesirable. It appeared to him that in whatever way they looked at the papers before them, the moral was that dry crushing should be left severely alone. There was friction, wearing out of parts, and nothing worked right from beginning to end. He was, therefore, strongly in favour of wet crushing.

He should like Dr. Simon to tell him where there was a mill that did not make dust.

Dr. Simon said that there were two known to him; one was at Hill End, Broad Arrow, and the other at the Norseman.

The President explained that Dr. Simon was referring to ball mills—not stamps.

Dr. Simon observed that Mr. Rathbone could see such ball mills at work both in England and on the Continent.

Mr. Rathbone remarked that he was referring to dry stamping.

Dr. T. Kirke Rose said that he could not agree with Mr. Merricks's remarks on the roasting of ores containing tellurium. Mr. Merricks stated that in roasting gold ores containing tellurium, there was a risk of loss of the precious metal by volatilization. He was aware that that belief was very general, but he did not think it was well grounded. Experiments he had made showed that, as near as could be ascertained, absolutely no gold was volatilized, but that the compound formed by heating gold with tellurium at a roasting temperature soaked through the ore and was taken up by almost any furnace bottom, so that it was difficult to avoid loss. In roasting tellurium ores, attempts had been made to recover the volatilized gold in flues, but the experiments he had made (which were published in the Report of the British Association last year) tended to

show that no gold could be recovered under such circumstances, and that it had to be looked for below, and not above.

With regard to the question of the tellurium being dissolved by cyanide, and precipitated by zinc, he wished to point out that if tellurium was alloyed with gold and treated with cyanide, it had a quickening effect on the dissolving action of the cyanide. The gold appeared to be positive to the tellurium, and was dissolved more rapidly in consequence of the presence of the tellurium, and none of this element was attacked as long as any of the gold remained undissolved in contact with it.

Mr. A. G. Charleton inquired if any of the members had had any experience with shelf drying kilns, as he understood they had been extensively introduced in America, and one would like to know how they compared in efficiency and economy with rotatory "driers."

With regard to dry crushing, Mr. McConnell stated that the discharge power of a dry stamp was entirely due to the concussion of the blow which threw the crushed ore against the screen, only portions of which passed through, the residue falling back into the box; he gathered from that statement that an exhaust was not being used for dry crushing, and he would have thought that by exhausting, an improved "duty" would be obtained. He considered, however, that there was another direction in which improvements in dry stamping might be looked for if certain mechanical difficulties could be overcome, viz., instead of exhausting air from the boxes, to force air under pressure into the "mortar" and force the ore out. A plant was being erected in London to give this system a thorough trial.

The duty of wet crushing stamps in New Zealand appeared to be extremely low, but he expected that it was due to the fact that, using cyanide as the writer of the paper described, they had to be very careful in economising their solution, and consequently could not employ enough water to carry out the ore rapidly enough for fast crushing. The author had not touched upon the question of the consumption of cyanide; he should have thought that, owing to agitation, the consumption of cyanide would have been considerably increased, and would have to be debited against the improved extraction obtained by that method.

In the concluding paragraph of Mr. Merricks's paper he noticed the proposal to hand pick the ore and send it to the smelters. That had already been tried, and as far as he was aware most of the companies who had done so had abandoned the plan, as, owing to the excessive cost of shipping and the charges for treatment at the

smelters, they found it would be more profitable from a commercial point of view to deal with the ore on the spot by dry crushing, roasting, and cyanide.

He was quite as enthusiastic as Mr. Rathbone as to the advan tages of wet crushing until he visited Kalgoorli. Whilst there, some experiments were made by Mr. Mariner at the Boulder Main Reef which seemed to indicate that if telluride ore was crushed wet with the saline water which had to be used in this locality, and it was afterwards roasted, the telluride caused a volatilisation of the gold. Dr. Rose had probably not experimented with ores with saline waters such as had to be dealt with in Western Australia, and it seemed that 40 or 50 per cent. of gold was lost by volatilisation under the conditions he had mentioned. If that were a fact, it would necessarily put wet crushing outside the sphere of practical politics in that particular field, and the ores of this district, generally, would have to be dry crushed as most of the companies proposed to do. On the other hand, with ordinary free milling stone, wet stamps were decidedly superior as crushers to dry stamps of any kind. These facts well illustrated the importance of adapting the method of treatment applied to any particular ore to the exigencies of local requirements.

Mr. H. L. Sulman said that, at first sight, it appeared from the figures given by Mr. McConnell as if there were but little difference between the consumption of cyanide by the agitation process and that which was lost by percolation methods. Closer consideration showed, however, that whilst by percolation there was a loss of about 0·15 per cent. KCy as against an agitation loss of 0·1 per cent. KCy or a little over; in the former case the loss was spread over a period of 70 hours; the agitation loss occurred in 16 hours. In the further tables given a still reduced agitation period of from 7 to 11 hours was yet attended with a constant consumption of over 0·1 per cent., which was a high amount. It appeared therefore that the figures once more bore out the old contention that agitation consumed more cyanide than ordinary percolation, time for time. Moreover, the actual loss of cyanide seems to have escaped due attention, and to be a feature in the treatment of many New Zealand ores. An average loss of 0·11 per cent. in the strength of the liquors, when these too were used in the proportion of 2 : 1 on the ore, meant a destruction of 5·0 lb. of cyanide per ton of ore treated. This was excessive in comparison with cyanide practice elsewhere, being, for example, from 10 to 15 times as great as average Rand losses.

They had to consider the subject, not with regard to the general

proposition of "*wet* versus *dry* crushing," but as to its bearings on the class of ores found in New Zealand. These constantly presented the peculiar characteristics of extreme dissemination of the gold and its association with an excess of sulphide of silver; both these factors called for special treatment and the devising of methods outside the ordinary lines.

Though he cordially agreed with most of Mr. James's remarks, he must take exception to his statement that the necessity for agitation was due to the larger size of the particles produced by wet crushing. The undisputed fact that in many cases it was necessary to crush to 90-mesh in order to obtain high extractions, clearly showed the great difficulties in the way of ordinary percolation when applied to these ores—in fact, *any* ore crushed to 90-mesh would be exceedingly difficult to leach. It was probably rather for the overcoming of the difficulties, due to the extreme fineness of the ore particles, that agitation had been advocated by Mr. McConnell and others.

He thought Mr. McConnell hardly gave them enough information as to the manner in which the ore, after it passed the battery, was dealt with. He would like to know how the finely divided ore residues deposited in the settling tanks were drained and washed, and also what depth of deposit could be dealt with on the vacuum filter beds.

One of the previous speakers drew attention to the considerable difference which existed between the results obtained by mere assay of ore and tailings and the amounts of gold actually recovered. It would appear that unless very complete drainage and washing took place on the filter beds, the tailings must retain amounts of cyanide and gold solutions, which would considerably add to the "leakage" not shown by assay results.

Referring to Dr. Rose's observations as to losses in roasting ores containing telluride of gold, he had himself carried out a series of experiments which pointed to another and additional source of loss. He found that when the readily melted globules of rich tellurides were exposed to an oxidising atmosphere, they rapidly became coated with a layer of highly insoluble tellurous acid, which, though thin, was almost like a glaze of porcelain over the small beads of partially roasted telluride. This layer not only prevented complete oxidation, but, being insoluble in dilute acids or alkalis, effectively protected any reduced gold from solution in cyanide, &c., subsequently.

Mr. A. C. Claudet remarked that the bullion produced by the Waihi Mine contained on an average about three times as much silver as gold, which would tend to show that the sulphide of silver

was attacked considerably by the cyanide, hence the low grade of bullion. A careful chemical analysis showed that there was a small amount of tellurium and selenium present in the bullion.

With regard to drying the ore, he understood that shelf kilns were used on a large scale in Borneo and the States. He thought that some idea of the temperature should be given when the term "drying" was used—was it simply sufficient heat to dehydrate the clayey material?

With regard to the classification given in the tables, it seemed that, allowing for errors and variations which were impossible to avoid, there was no practical difference at all between crushing dry or crushing wet through a 30-mesh screen. The figures in the two tables showing the grades of size obtained by stamping through a 30-mesh screen in the case of wet and dry crushing compared as follows:—

	Wet.	Dry.
Left on 40-mesh	1·0	1·2
„ 60 „	8·0	9·0
„ 90 „	16·8	12·56
Passed 90 „	74·2	77·24
	100·0	100·00

He thought that those figures showed that there was practically little difference between the two. He understood that the experiments had been conducted on a large scale, and was very much surprised that the loss of cyanide was so small.

Speaking generally with regard to wet *versus* dry crushing, there was no doubt that if stamps could be avoided for dry crushing, it was better to do so. He thought that, in investigating what could be done with an ore and how it would act under cyanide treatment, a systematic series of experiments should be made; for instance, the ore should be crushed through ¼-in. holes, then graded down, and each grade assayed for gold, and the weight of each grade given. Each grade should be cyanided separately, and he considered that such experiments would be of great use in deciding what machinery should be used for crushing. He was of opinion that dry crushing by stamps would gradually become a thing of the past. It seemed to him that the loss of cyanide with wet crushing must be greater than with dry crushing, because in the latter the ore was simply crushed, passed into a vat, and the solutions run off. What happened in wet crushing? There were solutions circulating all over the place, and he had also found that cyanide solutions rapidly deteriorated simply from contact with the atmosphere, and therefore

the more the solutions were circulated the greater was loss of cyanide from this cause.

He was in favour of dry crushing, but not with stamps, and direct cyanidation, especially on certain low grade complex ores. He was connected with a company in New Zealand where they had a large lode, between 12 ft. and 20 ft. wide, with an extremely big body of ore, which was very porous and yielded its gold very readily to ordinary coarse crushing to ¼-in. holes mesh with cyanide solutions direct.

Mr. Cordner-James said that, in reference to Mr. Claudet's remarks, experiments had recently been carried out on an ore, assaying 8 dwt., passed through a 30-mesh screen, with relative grading to 40, 60, 80, and 90, which gave results showing a progressive increase in value for the finer grades of ore.

He had made considerable experiments in connection with the roasting of tellurides, and his results bore out Mr. Charleton's remarks on the subject. His experiments had shown that with the water obtainable in Western Australia, using wet crushing and roasting the concentrates, the losses had varied from 15 to 25 per cent. Telluride ores could be roasted, after dry crushing, when common salt and other chlorides were not present, with losses averaging from 1½ to 2 per cent., and he had been able to roast them on a fairly large scale without exceeding these figures.

He was obliged to Mr. McConnell for his paper, and it was quite evident that each class of ore had to be considered on its own merits as to whether wet or dry crushing should be adopted.

Mr. Nicol Brown said dry crushing was started at the Lisbon Berlyn Company's Mine because of the large percentage (about 40) of slimes produced by the wet stamping.

When first started, the ore taken from near the outcrop, being comparatively dry, was more easily dealt with by the dry crushers than more recently when the ore has been drawn from farther into the hill, and consequently carries more moisture.

They had overcome the difficulty to some extent, but it was evident that the results would be very much better if the ore were drier, as in dry weather the results seemed to be more satisfactory than in wet weather. Unfortunately, artificial drying was out of the question from want of fuel. No doubt Mr. Matthew Taylor Brown, who was present, would be able to explain the process in use at the mine he had referred to.

Mr. Matthew Taylor Brown, said the Lisbon Berlyn Company's ore was originally dealt with by the ordinary wet crushing and amalgamation process, but with ore of, say, 8 or 9 dwt.

contents, only about 1 to 1½ dwt. was caught on the plates, and about 40 per cent. of slimes was produced.

Dry crushing and direct cyaniding were introduced at first in a tentative way as there was no experience with similar ore at other mines to rely on. The first trial was made by screening off all the smalls coming from the Blake rock breaker, which would pass a $\frac{3}{8}$ or $\frac{7}{16}$ in. bar screen. This quantity was found to amount to about 30 per cent. of the whole ore crushed, and it was taken direct to the cyanide vats, the balance continuing to be treated as before in the stamp battery.

The success met with justified the extension of the direct treatment method, and two Marsden fine crushers were introduced to deal with the coarse product from the rock breakers, so that all the ore might be dry crushed.

These fine crushers, when the jaw faces were new, were able to reduce the ore to about the size of peas, which was as fine as was wanted, but the corrugations quickly wore off, and a much coarser product was the result after the first few days. To crush fine enough meant replacing the movable jaw faces every three or four days, which was a very heavy expense.

This difficulty has been got over by adding a set of Gates rolls as a third stage in the crushing operations, and these rolls now give a satisfactory product. The shells of the rolls need to be turned up only about every six weeks, and the Marsden fine crushers instead of being set to give a very fine product are now an intermediate step in the crushing process, and the jaw faces which had been discarded, only partly worn, can now be put back into the machines and more duty got from them.

Briefly, then, this arrangement as at present in use is as follows:—The crushing is done in three stages, by Blake rock breaker, Blake-Marsden fine crushers, and Gates rolls. Between, or after each stage, the ore is screened or trommelled, so that what is already small enough does not pass on to the next stage, but is shunted off instead to the vats.

When the direct treatment was first introduced, the ore was very porous, and a coarser product was permissible than latterly, and the necessity for the three stages of crushing was not then so obvious.

As the ore is drawn from further into the hill, it becomes damper, and it is found that clayey or earthy matter adheres to the surfaces of the crushed ore and renders the particles less permeable to the liquor. Washing the ore before crushing has been suggested as a method of getting rid of the adhering envelope of earth or clay, and may prove effective.

The extraction over the last 12 months has been 68 per cent. on a filling assay of about 10 dwt., but this we hope to improve on, and the theoretical extraction and actual recovery closely agree.

The gradual adoption of the process not only admitted of feeling one's way in an untried direction, but prevented the necessity of a lengthy shutting down of milling operations during the transition period.

The President, in closing the discussion, observed that he thought they could draw from the remarks which had been made, the conclusion that the conditions differed so much, and the character of the ores was so varied, that no rule could be laid down as to the treatment of the New Zealand ores. Mr. Brown had given them particulars of a case where coarse crushing and cyaniding direct had been profitable, while Mr. James and Mr. McConnell had drawn attention to the fact that even the ball mills did not bring the New Zealand ores to a sufficient degree of fineness. The extraction, after crushing by ball mills, obtained from the New Zealand ores, had not been satisfactory, consequently it was not safe to assume that any coarser crushing would answer the purpose. If they examined the fineness of the product from dry crushing, as compared with wet crushing, there was generally a great difference in the production of fines and the discharge was not so easy as with the assistance of water. If they examined under a microscope the pulp obtained from different systems of crushing, such as rolls, wet stamping and dry stamping, they would find the particles from a dry crushing mill were rounded, from rolls they had sharp edges, and from wet stamps were not so completely rounded at the edges.

Mr. Claudet had called attention to the apparent discrepancies in Mr. McConnell's paper on his experiments with different quantities of slimes. Curiously enough, those results rather bore out the theory as to the difference between dry and wet crushing, by establishing the fact that as the discharge became more rapid the relative proportion of fines was lessened. With a 30-mesh screen, using 1·6 tons of water to 1 ton of ore crushed (which was a very small quantity of water as compared with the ordinary wet stamp process), 69·2 per cent. passed through 100-mesh, the rest being coarser. Stamping with 60-mesh and using 2·8 tons of water to the ton of ore, only 68 per cent. passed through 100-mesh—in other words, using a 60-mesh screen and twice the quantity of water did not make any more fines than using a 30-mesh. He thought that was a striking corroboration of the fact that the facility of the discharge diminished the amount of slimes produced.

It had been suggested that a sucker should be placed in front,

and compressed air behind, and so draw the stuff out quicker, but if that were done the amount of fines would be diminished. The reason for continuing to crush through a 30-mesh screen in New Zealand appeared to be that a greater extraction and more finely crushed ore was obtained thereby; if it was required to crush so fine, efforts to increase the discharge of the stamp battery would only result in having to put up a finer screen to keep the stuff in until it was crushed.

Mr. Rathbone made a very sweeping statement in regard to dry crushing, but subsequently modified it as only applying to stamp mills. There was no doubt it was extremely difficult to prevent the discharge of dust with stamp mills, and this was one of the objections to their use. Notwithstanding the utmost care in constructing such a mill, the vibration caused by the stamps made it very difficult to avoid leakage of dust, and therefore, if dry stamping was adopted, it was only on account of its being necessary to get the ore exceedingly fine.

With regard to the roasting of tellurides, he thought that experiments made in roasting particularly rich telluride ores did not necessarily apply to ores like those in Western Australia, where there was only a small percentage of telluride minerals scattered through the mass of the rock. Such an ore had at no one portion any quantity of telluride. One of the reasons which made wet stamping, as a preliminary to dry treatment, in Western Australia, out of the question, was the fact that the ore there consisted principally of bands of slate with quartz seams through them, and talcose and chloritic slates which were tough in character, and had to be very well dried if it was intended to stamp them, while even then it would be a very nasty material to put under a stamp for dry crushing. On the other hand it was well adapted for the use of ball mills, which were gradually being introduced into the country. There were, however, very decided disadvantages attaching to the use of ball mills, and on ordinary quartz they could not be compared to wet stamps, where it was possible to use the latter.

With regard to wet crushing and cyanide treatment, Mr. James had pointed out very fairly, that there was no reason why the separate treatment of the different products produced by wet crushing should not be carried out in the same way as on the Rand. As he had stated, however, 1·6 tons of water to the ton of ore was too small to get a proper duty. In America the ordinary gold milling practice on quartz would be about 4 tons of water to 1 ton of quartz, while on the Rand it was about 10 tons of water to 1 of quartz.

CONTRIBUTED REMARKS.

Mr. Edward Hooper: In dealing with this most important subject of dry *versus* wet crushing, Mr. Merricks fails to throw any considerable light thereon, in the shape of actual results which are capable of comparison, and his adverse criticism on ball mills as dry crushers is unsupported by any testimony of value. He admits that the "Waihi" Company successfully substituted dry crushing followed by direct cyanidation, instead of wet crushing and amalgamation, and charges the companies in the surrounding districts with having in consequence blindly adopted the dry crushing process: yet he prophesies that during the next year most of these companies will be, or ought to be, wet crushing their ore, simply because the Crown Company is converting their plant to wet crushing.

The precise nature of the treatment recently adopted at the "Crown" Mine has not been made public, but wet crushing experiments have also been carefully conducted on the ores of the "Woodstock" and "Talisman" Mines. The "Crown" ore is mostly clean quartz, and comparatively free from base metals and slimes, the value of the resulting bullion being 30*s.* to 45*s.* per oz. On the other hand the "Talisman" ore is sometimes much mixed with clay, and contains in addition to sulphide of silver, about 0·9 per cent. of iron and copper pyrites. The value of "Talisman" bullion is only about 17*s.* per oz. Apparently the wet crushing tests at the "Woodstock" Mine have not met with such enormous success, and the following notes on some experiments made with the "Talisman" ores may be of interest to the Members of the Institution, viz.:—

The ore was crushed wet, through 30-mesh screens, a 0·85 per cent. KCy solution being used in the mortar boxes. Two vats were charged at the same time with ore to a depth of about 20 in. After syphoning-off the first solution, a 1·1 per cent. KCy solution was used in vat No. 1, and 1·2 per cent. solution in vat No. 2. After allowing these solutions to percolate for 36 hours, the ore was treated with sump and water-washes, the time taken to complete the treatment being six days. The extraction results were as follows:—

	Assay before Treatment.			Assay after Treatment.		
	oz.	dwt.	gr.	oz.	dwt.	gr.
Gold.....	0	14	16	0	2	10
Silver	7	15	4	5	8	16

The total value extracted being 69 per cent.

The slimes (7 per cent. of the ore) which accumulated from the syphoned solution, assayed as follows per ton :—

	oz.	dwt.	gr.
Gold.....	0	8	4
Silver	8	4	23

Mr. Merricks, to support his views against dry crushing, makes the sweeping assertion that it has been abandoned on the Rand in favour of wet crushing, without, however, seeking to discover the reason thereof.

The early use in New Zealaud of crudely constructed kilns for the purpose of drying the ore is of course to be deprecated; but this method is now being superseded by the use of revolving dryers. One of these dryers is now used in the mill of the "Talisman" Company, with the result that from 1 to 3 tons of wood fuel are now used in drying 15 tons of ore, as compared with 1 ton of wood fuel to 2¼ tons of ore in the old kilns.

A certain divergence of opinion undoubtedly existed in the early days as to whether dry or wet crushing was the better and more economical for treating the Kalgoorlie ores of Western Australia; but at the present time, or rather since the discovery of tellurides and sulphides at a depth of from 100 ft. to 200 ft., it is generally admitted that a dry crushing plant is the proper one. Even the oxidised ores in this district vary considerably in their mineral constituents, for the "Ivanhoe" and "Lake View" ores are much more quartzy than those occurring in the "Brownhill" mine. The oxidised ores in this district, which have been treated by wet crushing and amalgamation over plates, only yielded about 50 to 60 per cent. of their gold contents, and the resulting rich tailings contained a very large percentage of slimes. It is of course not practicable to treat these slimes by the ordinary cyanide method, consequently two separate plants are necessary for extracting the gold from the sands and slimes.

In concluding, Mr. Merricks advocates the following method of treating the unoxidised ores of the Kalgoorlie district, viz. :—The high-grade ore, consisting of the rich tellurides, to be handpicked for shipment to smelters, and then to wet crush and amalgamate the low-grade ores, saving any remaining tellurides by concentration, and shipping same to smelter; the tailings, if rich enough, to be divided into sands and slimes, the former being treated in the ordinary cyanide vats, and the latter being passed through filter-presses. This appears to be far more complicated and expensive than the dry crushing methods now being adopted on these fields.

Moreover, the practicability of saving tellurides by concentration appears somewhat doubtful, not only on account of the scarcity of water, but also, to quote from a paper written by Mr. W. P. Ingalls in the "Mineral Industry" for 1897, that "finely crushed ore, containing its value in the form of brittle telluride minerals, appears ill adapted to gravity concentration."

Mr. J. McConnell in his "Notes on Dry and Wet Crushing with Cyanide Treatment in New Zealand," refers presumably to experiments made with the ores of the "Woodstock" Mine. Seeing that the ores from this mine have been treated by the wet crushing process for several months past, the question as to whether this or the dry crushing process is the best, should have now passed beyond the experimental stage. Mr. McConnell's paper, however, deals principally with the respective merits of cyaniding wet crushed ore by the ordinary percolation method or by agitation, and concludes with the remark that with the Woodstock ores, "an agitation treatment is necessary for a high extraction." It would be very interesting to know the cost of the dry crushing process used in the past, as compared with the wet crushing method now adopted. The statement made by Mr. McConnell that "the actual consumption of cyanide per ton of ore treated (wet) is as near as possible equal to the consumption in the treatment of dry crushed ore," is one which has been partially verified in some wet crushing experiments on "Talisman" ore, where the loss amounted to nearly 3 lb. KCy per ton of ore treated. It would certainly be thought that in using a cyanide solution in the mortar boxes, the iron would be partially dissolved, and have a deteriorating effect on the solutions; besides which, some of the metallic mercury would probably be dissolved by the cyanide.

Mr. F. Merricks: In regard to Mr. James's remarks on my paper, I think he must have misread or misunderstood what I intended to convey in it.

In the first place I entirely fail to see why or where Mr. McConnell's paper very much answers the suggestions raised or made by me.

As to the output by wet crushing the Ohinemuri ores, I see no reason to alter my estimate of 3 tons in respect of this. I do not wish to imply that this would be the case by crushing in the haphazard sort of way that has, until the last few years, been so much in vogue in New Zealand practice generally; but given a modern mill, with appliances such as exist on the Rand, I certainly think the above results would be obtained. I am supported in this by the fact that in the extensive experiments made at the Crown Mine, the output

has, with a very small and limited amount of water, been increased to 2 tons per stamp per day, and, further, one of the points aimed at in these experiments is the treatment of the slimes, which are run off and separately treated and not all run together coarse, fine and slimes into one vat and treated together, as appears to Mr. James. In regard to these experiments and considering that no results of such have been published, I think I took only a practical and common sense point of view when I assumed that they must be considered satisfactory, since the whole plant has been converted to wet crushing, which fact would make it appear that they are not experimenting in this respect, and as wet crushing experiments have been and are still being conducted by other companies, I think, probably, by the end of next year I shall have the satisfaction of knowing that my statements have been justified by results.

As regards my statement of the various companies adopting dry crushing without considering the advisability of trying raw wet crushing and cyaniding, which Mr. James appears to have overlooked, it is certainly news to me to know that almost every mine at work in the district originally tried this course unsuccessfully. I am perfectly well aware, and as I state in my paper, the Ohinemuri ores cannot be treated profitably by wet crushing and amalgamation alone, but what I wished to imply was, that by carrying the treatment a stage further and subjecting the tailings to cyanidation (without passing cyanide solution through the boxes), the results obtained, if not quite so satisfactory from an extraction point of view, would be more than counterbalanced by, or as I contend, the increased output, and would overcome the difficulties pointed out by Mr. James.

I am perfectly willing to admit that some of the Karangahake ores are not suited to this method of treatment, for, as will be gathered by studying the assay value of the ores experimented on by Mr. McConnell, it will be seen that the gold and silver bullion contents of such are as 1 to 15, and should be classed as a silver ore rather than a gold ore. There are, however, other ores in the same and surrounding districts of a much higher bullion grade, and such I would suggest as being suitable for wet crushing, and do not require to be crushed so fine as the particular class of ore referred to by M. McConnell, for instance, at the Silverton Mine, where the bullion is worth about £2 an ounce, dry crushing with a 20-mesh has been used for the past two years with extraction results of from 50 to 65 per cent. of the silver and about 90 per cent. of the gold.

Mr. James says I "quietly assumed, without giving any figures or data, that because with an exceptional ore, which was perhaps

the one ore in the world best adapted to the process (the Rand 'Banket' ore), 5 or 6 tons per head per diem could be crushed and treated by amalgamation and cyanide, it could also be done on an exceptionally difficult ore such as was found in New Zealand," such, however, is not the case, as is clearly shown by the following paragraph contained in my paper. "The author is of the opinion that the improvement effected at the Crown Mine should be completed by adopting wet crushing, amalgamation, and cyaniding in its entirety. By this means the output would be greatly increased, say to 3 tons."

In presenting my paper to the Institution, I had hoped that it would have brought forth a more general discussion on the subject, at the same time, I am glad to know that it may be the means of inducing other members, with large experience in dry crushing in the Transvaal and West Australia, to present papers to the Institution.

A paper communicated by Mr. Alfred James, entitled "Notes on a Process for Treating Slimes without Filtration or Decantation," was, owing to the lateness of the hour, carried over to the next meeting.

THIRD ORDINARY MEETING, 21st December, 1898.

Mr. WALTER McDERMOTT (*President*) in the Chair.

The minutes of the previous meeting were read and confirmed.

DISCUSSION—*continued*, "On Dry and Wet Crushing" (JOHN McCONNELL and FRANK MERRICKS).

The President said that at the last meeting a long discussion took place on two papers relating to dry crushing as compared with wet, and a variety of opinions were expressed, with warm advocates for both systems of treatment. The general concensus of opinion appeared to be that the peculiar circumstances existing in Western Australia made the adoption of dry crushing possible and advisable, whereas the same could not be said as to the conditions existing in many other countries where the two processes had been tried. Some opinions were also expressed as to what had been done in that direction in South Africa, and he thought that on the whole the impression was, that they were not really in possession of all the facts, and that there was a tendency to condemn it as a whole. Mr. Franklin White, who was a recognised authority on the subject, and who had read papers in South Africa in connection therewith, was present that evening, and could no doubt give them a considerable amount of information as to the relative advantages of the processes, and the actual results obtained in the Transvaal.

Mr. Franklin White thanked the President for giving him an opportunity of speaking on the subject, which was a most fruitful one for discussion. They had had some very interesting, and at times, rather warm debates on the question in South Africa. There were a few enthusiasts there who could be found to champion the cause of dry crushing, but the opponents were very numerous. On reading through the paper which was discussed at the last meeting, it seemed to him that the author hardly realised his aspirations of saving many a mine manager worry and expense in deciding

whether to adopt wet or dry crushing. The paper was remarkably free from any data or facts which would assist managers to decide whether his ore was suited or not to one system or the other; it seemed to be principally a condensation of data which had been acquired by the author, who did not give many reasons for the opinions stated—if more facts and figures had been given, the paper would have been of greater assistance. Personally, he was not prepared to make any remarks on the system of dry crushing as carried on in New Zealand, as the suitability of it depended very much upon the ore, the class of gold, and the local conditions. He noticed, however, that the author referred to "crudely constructed kilns," and he considered that ought not to be regarded as a very valid objection to dry crushing, because, if it were worth while, surely properly constructed kilns could be made.

He was most interested in the very decided statement that dry crushing on the Rand had been abandoned in favour of wet crushing; he took it that what was meant was that dry crushing plants had been tried and superseded by stamps. He knew pretty well most of the plants where dry crushing had been tried on the Rand, and as far as he was aware, not one of them had been superseded by stamps. It was possible that the majority of them were not working at the present time, but their suspension depended upon many considerations, possibly of finance, or pending the decision of the companies for the future working of their properties. He did not feel called upon to say whether the works were as perfect as they might be, or whether the results obtained were satisfactory, but the general impression was that they, the parties most interested, were satisfied with the work done in the plants at their disposal. Most of the plants were experimental ones, and it was not pretended that they were perfect, but he believed he was right in saying that not one of them had been condemned as being deficient in the qualities which the constructors expected. Sundry machines had failed, but that did not affect the principle.

The first experiments he made were on the mineral belonging to the Village Main Reef Company, and as far as he personally was concerned, he considered they were satisfactory; but a change of management took place, and the new engineers considered—possibly they were wise—that with ore of the value of the Village Main Reef, which left a considerable margin of profit, it was wiser and safer to adopt the old well-known methods, and not risk any danger of criticism by taking up new processes. If such conservative ideas were always followed, there would be little hope of improvements taking place. Naturally, those who had to design plants for rich

ores, if they followed out such conservative notions, would stick to the old fashion, while those who had to try and make lower-grade mines pay, would look about for a cheaper plant and a cheaper mode of working, by adopting dry crushing. The result would be, that on the one side there were the richer mines working with wet mills, and on the other, the poorer mines working on more or less experimental lines.

At the Luipaard's Vlei Estate property, of which he had been the consulting engineer, a fairly substantial plant had been erected which, up to the time of his leaving, had treated some 25,000 tons of ore, which could hardly be called an experiment. It would be too long to enter into details that evening, but he would be pleased to read a special paper on what had been done at that mine, as a guide to other managers, and to show them what could be done on that class of ore. In order, however, not to fall under the criticism which he had meted out to the author of the paper—giving opinions without facts—he would give them a few of the figures.

He found that in wet stamp mills the cost of wear in stems, heads, shoes, and dies, was 3·1*d.* per ton crushed, and the wear of material was equal to 0·72 lb. per ton. That did not include the wear of the stems. Experiments at Luipaard's Vlei had shown that the wear and tear, which was supposed to be so great in dry crushing, was 0·80 lb. as against 0·72. He had two rolls and a Niagara pulveriser. Owing to the makers of the latter not having hit upon the proper hardness of the material, they had to throw away about two-thirds of the weight of crushing surfaces before they were worn out. The cost was certainly higher; their costs were 5·4*d.* per ton, which was not so terrible as compared with 3·1*d.*, seeing that they were able to work with fully 20 per cent. less horse-power, and the saving in the fuel bill would more than compensate for the little extra wear and tear.

Stamp work has attracted the attention of a great many clever men for a long time, and they had got as near perfection as possible, and he hoped that in two or three years' time manufacturers would pay more attention to dry crushing requirements, and give material better suited as a result of the experience gained.

Another point in favour of dry crushing was the small amount of water used. In a wet mill at least seven times as much water as ore had to be used, and the water had to be raised to the tanks which supplied the battery, and afterwards raised again by the tailings-wheel elevators. Dry crushing saved that, and they were enabled to get through their cyaniding with 24 gallons to the ton of ore—240 lb. to the ton. As to the capacity of the plant, they were

able to crush through a 300-mesh at the rate of *9 tons per hour*. The plant was really equal to a 40-stamp mill of heavy pattern. The total horse-power used in running the three crushers, belt conveyers (which carried the material from one part to another), the elevator (which raised the ore to the screen floors) and the screens, was 86, which was much less than would be required for a 40-stamp mill with its pumps and tailings elevator. With regard to extraction, 200-mesh gave about 85 per cent. of the assay value, which, seeing that they were able to get a perfect assay of the ore going into the cyanide tanks, was a very fair practical result. It was possible that the extraction might have been increased, but he doubted whether the profits would have been larger. He attached a good deal of importance to assays. The unaccounted for only amounted to 1 per cent.: he thought if they had scraped their flues they might have decreased even that 1 per cent. As it was, they acknowledged the unaccounted 1 per cent., which was not usually done by mines accountants, who generally managed to square their accounts, so that there was apparently absolutely no loss. This he thought was due to the careful work of the secretary rather than to the actual work on the mine! They were still experimenting when he left, and the costs were naturally higher than they would be in a mine which had been running for some time on a settled basis. This had to be taken into account when passing judgment on costs. Some of them had possibly put up experimental works, and knew the position the manager was in when he found he had made a mistake, and spent money which, all the same, had to go against the cost.

To those using dry crushing, it seemed that the stamps crushed the ore finer than was necessary, whereas with rolls the size could be regulated to a certain extent. Another point in connection with stamps was the amount of horse-power required. To crush 1 lb. of ore they had to do an effective work of 8,000 ft.-lbs.

Mr. H. K. Picard thought that, in considering wet and dry crushing, it had to be taken into account what was to be done with the ore afterwards. He was of opinion that for ore that did not require expensive water-washing, the dry process was better, but on ores where washing was necessary, wet stamping was vastly superior.

The President, in closing the discussion, said that several speakers had made the mistake of attempting to judge broadly between dry and wet crushing, when there was really no comparison possible, as there were conditions which made one method necessary and the other impossible.

"Notes on a Process for Treating Slimes without Filtration or Decantation."

Communicated by ALFRED JAMES, M.Inst.M.M.

IN bringing these notes before the Institution, I must explain at the outset that my doing so must not be taken to imply any approval or recommendation of a process of which I have no personal experience. Credit and criticism alike are due to Messrs. Gilmour and Young, who have worked out the method and brought it to the successful issue detailed herein, and now that it has passed beyond the experimental stage and has treated thousands of tons of ore and produced over 10,000 oz. of bullion, I feel that notwithstanding such apparent objections as the difficulty of complete solution and precipitation in a viscous or semi-fluid mass, the total destruction of the cyanide used, the heavy expense for chemicals and horse-power, and the baseness of the bullion produced, the process has proved its applicability to practical use, and that it has in fact proved the salvation of a mine the total output of which has been treated by it for the past 12 months. Therefore I am passing on to the Institution the records sent to me, and do so that members may be in a position to try for themselves whether the process may not be the solution, in some cases at any rate, of what at present is acknowledged to be a difficult and expensive problem.

This method has been employed for the treatment of the ore of the Santa Francisca Gold Mining Company, Nicaragua. This ore contains a very large percentage of clay, which renders leaching a matter of extreme difficulty. The ore as a whole can be treated by the aid of a vacuum in about seven days, but the slimes, which amount to 70 per cent. of the ore, are not satisfactorily leached and washed even after 30 days percolation.

The ore is crushed by a No. 5 Krupp mill through a 30-mesh sieve, and it is found that the slimes carry more gold than the sandy paticles.

Until recently the pulp was treated by amalgamation in an

ordinary boss pan plant for an extraction of 50 per cent. of the gold and 38 per cent. of the silver, but the rich slimes lost but little of their gold contents by this treatment. The plant consists of four Boss pans, 5 ft. diameter, and two separators or settlers 7 ft. 6 in. diameter, and is used without alteration for the new proccess.

Amalgamation was superseded by cyanidation, and the San Luis ore ground through 30-mesh gave an extraction of 80—85 per cent. of the gold, and 50—60 per cent. of the silver for 6—7 days treatment, at a consumption of 1¾ lb. cyanide per ton.

The dry crushed ore is charged into a pan in 2-ton lots, with 100 gallons of water so as to form a very thick pulp. From two to six bottles of mercury are added, until the globules can be seen circulating in the pulp, and then the required amount of cyanide. After running this for about two hours, 10 lb. weight of mixed zinc and copper amalgam is added and the pan run for four hours longer. By this time the precipitation of the gold from the solution is stated to be very complete, only 10 gr. of gold per ton remaining in solution, and this has even been reduced to 1 gr. per ton of ore. This shows a great advantage over the decantation process, where, owing to the large amount of water used, the assay of the treated solutions has to be multiplied by 8 to show the loss on the ore, unless a portion of this gold is saved by the re-use of the solutions.

The pulp and solution are then discharged into a settler and the mercury recovered as in the ordinary pan process. By re-using the amalgam in larger quantities as the base metal contents are reduced, bullion of about 800 fine is obtained. It is found that although one hour is sufficiently long for the solution of the gold in the slimes, the sands may require longer treatment—say 4 days percolation, and are then advantageously separated by spitzkasten from the slimey pulp, and given separate percolation treatment. The slimes show an extraction of over 90 per cent., and the sands, without re-treatment, 75 per cent., an actual bullion recovery of 84 per cent. on the whole ore. Re-treatment of the sands brings the extraction of the gold up to over 90 per cent.

The slimes are now being air separated, and are yielding an extraction of 90 per cent. of the gold and 70 per cent. of the silver when run with about 50 per cent. of water. These slimes assay—

Gold.			Silver.		
oz.	dwt.	gr.	oz.	dwt.	gr.
1	10	0	4	10	0

Appended hereto are statements of results of ore treated and cost of chemicals, horse-power, and labour required.

Result of Treatment of Santa Francisca Ore by this Process. For the twelve months ending June, 1898.

Date.	Tons.	Charging assay.		Metal in charge.		Metal in bars.		Extraction.	
1897 to 1898.	2,000 lb.	Gold.	Silver.	Gold.	Silver.	Gold.	Silver.	Gold.	Silver.
		oz. dwt. gr.	oz. dwt. gr.	oz. dwt. gr.	oz. dwt. gr.	oz. dwt. gr.	oz. dwt. gr.	Per cent.	Per cent.
July	100	2 0 0	3 12 0	200 0 0	360 0 0	983 2 10	1,492 0 15	81·2	72·7
August	177	2 0 16	3 6 11	359 18 0	588 3 13				
September	314	2 1 11	3 10 6	650 17 22	1,102 18 12				
October	226	1 16 11	2 2 4	411 19 15	476 9 15	368 15 5	415 8 10	89·5	87·1
November.....	167	1 5 15	2 2 8	213 18 13	353 7 10	154 8 10	159 3 20	72·3	45·0
December.....	292	0 18 20	1 19 3	443 4 5	921 5 12	353 13 15	517 0 15	89·3	62·9
January	471	1 9 21	2 6 15	703 4 0	1,097 18 0	978 12 5	1,318 11 0	87·2	70·5
February.....	390	1 1 12	1 19 12	419 5 0	770 5 0				
March.....	573	0 18 7	1 18 22	523 14 10	1,115 1 5	441 6 0	683 14 0	84·2	61·3
April.....	460	0 19 1	2 1 17	437 18 10	959 2 0	361 10 20	535 0 10	82·5	55·7
May	455	0 18 15	2 2 16	423 3 0	970 10 5	363 2 10	455 5 20	85·8	46·8
June	504	0 18 21	2 4 6	475 5 0	1,110 9 0	306 12 5	498 11 20	Part only recovered.	Part only recovered.
	4,129	1 5 12	2 7 14	5,262 8 13	9,825 10 0	4,311 3 8	6,074 16 14	83·6	63·9

The extraction is calculated on the bullion extracted and recovered in bars.
Re-treatment of sand brings gold extraction over 90 per cent.

The copper sulphate and cast-iron turnings are used to make the copper amalgam in a small separate pan. The zinc amalgam is very easily made by pouring mercury into molten zinc. It is advisable to keep the zinc amalgam under water to prevent rapid decomposition.

Chemicals used and Cost at Nicaragua.

	Lb.		s.	d.			s.	d.
Cyanide	1·67	at	1	3	per lb.	.....	2	1·05
Caustic	0·63	,,	0	1	,,	.....	0	0·65
Copper sulphate	1·16	,,	0	1¾	,,	.....	0	2·03
Cast-iron turnings	0·25	,,	0	0½	,,	.....	0	0·12
Zinc	0·39	,,	0	3	,,	.....	0	1·17
Mercury	0·40	,,	1	9	,,	.....	0	7·81
							3	0·8

These costs are for rich ores yielding about 3 oz. per ton; with ordinary tailings or slimes, the consumption of reagent is much less.

On slimes the consumption of cyanide is reduced to under ½ lb. per ton (Note.—African practice shows the consumption can be reduced to much less than this figure).

Labour comes to 3*d.* per ton.

Horse-power, 24 H.P., is required for 50 tons treated per diem. This would be much reduced by the use of barrels.

One of the advantages of this process is that the laborious cleaning up and melting of the gold-zinc slimes are avoided and the amalgam is simply retorted and dealt with in the ordinary way, and tests can be readily made in a pan or barrel at any time of the applicability of this method for the local slimes. The costs given above for the comparatively small daily output treated in a pan plant are necessarily high, but regarding this merely as a precipitation and cleaning up process, it appears to avoid the necessity for filter presses, electric precipitation, or handling gold slimes, and I shall be glad if these notes induce members to investigate for themselves the applicability of the method.

DISCUSSION.

The President considered that Mr. James's communication opened up a large field for discussion and experiment, as the process itself was simple in practice, and at the same time varied very little from other methods in which cyanide was employed.

Dr. A. L. Simon asked if Mr. James could inform them what was the chemical action of the copper amalgam which was added to the zinc?

Mr. H. L. Sulman thought that Mr. James's paper was a great stimulant to experiment and discussion, more especially as it was practically a return to what might be called the first principles of the treatment of slimes. Seeing that so many of the methods which had been tried for the treatment of slimes had turned out failures, the fact that in Nicaragua a method had been tried by which the addition of zinc amalgam and copper amalgam had been able to entirely precipitate the gold from cyanide solutions in the presence of slimes, came as a startling reminder that none of the best experimental methods could be considered perfect.

Mr. James rightly said that both credit and criticism in connection with the paper was due to the gentlemen who forwarded the communication to him; but he wished to ask one or two questions which might be passed on to these gentlemen. It was stated in the paper that the ore, as a whole, could be treated by the aid of a vacuum in about seven days, but the slimes (which amounted to 70 per cent. of the ore) were not satisfactorily leached and washed, even after 30 days' percolation; it appeared to him that in that case percolation could not be said to have taken place at all, as they were told in another part of the paper that the gold was in solution without any contact with cyanide after a period of about one hour.

Mr. James observed that he meant that the stuff was so slimy that it would not percolate.

Mr. Sulman, continuing, said that the consumption of cyanide was stated to be $1\frac{3}{4}$ lb. per ton, and the ore was said to contain about 70 per cent. of slimes, but, later on, they were told that with ordinary tailings or slimes the consumption of reagent was much less. He would like to ask whether the figures given in the table of chemicals used, and the cost at Nicaragua, referred to the total ore, *i.e.*, ore which had been treated as to 30 per cent. sand by percolation and 70 per cent. by the process of precipitation with zinc and copper amalgam. Did the results of the treatment of Santa Francisca ore include the whole ore, or did they refer to the slimes themselves?

Mr. James said that it had recently been discovered that by retreating the sands they could increase the extraction. Hitherto they had been treating the whole as slimes, until they found that by separating the 30 per cent. of sands they got a better extraction.

Mr. Sulman was of opinion that one was inclined to re-echo the question asked by Dr. Simon, as to the advantage obtained by using

copper amalgam. He should like to know how it was that zinc amalgam would not do the work itself, and why it was empirically found—as he presumed it had been, seeing that 10,000 oz. of gold had been recovered by the method—that the mixture of copper amalgam was also necessary?

The total cost, exclusive of labour, appeared to be 3*s.* per ton. He congratulated the authorities at the mine on being able to work at such a low figure as 3*d.* per ton for labour, and wished to know whether that included white supervision? It struck him that, with such low costs for labour, other methods would be just as likely to prove the salvation of the Santa Francisca Mine as the particular method described by the author. Figures were now to hand with regard to the filter-press method in use in Western Australia, and in a comparatively small press—holding about 1 to 1½ ton of slimes per charge—notwithstanding the high figures for Western Australian labour (which varied from £3 to £4 per week), the slimes could be efficiently handled and washed and the cyanide solution obtained from them for well under 3*s.* per ton. Applying this method to countries where labour was cheaper, the costs would certainly come down to something like 1*s.* 6*d.* per ton, and there was no reason why slimes should not be handled on a large scale at even 1*s.* per ton, when makers produced presses holding 3 tons per charge. He believed that if filter pressing had been used in Nicaragua on the scale it had been used in some of the West Australian mines, they would not only have had salvation at the Santa Francisca, but would have had 18 pennyworth more salvation in addition!

Mr. Claude Black asked if Mr. James could give any particulars about the drying of the ore. It was an important question, as in many parts fuel was scarce, and the cost of drying might be almost equal to the cost of the whole working?

He thought that the method had a great advantage over other cyanide processes, as there were practically no bye-products.

Nothing was said as to the composition of the ore, except that it was clayey. It would be interesting if Mr. James could give a more definite description. He knew many mines in Central America and the north part of South America where the ore was more or less of that character: there were very large quantities of decomposed felspar and kaolin in the ore which resisted amalgamation treatment, principally on account of the very finely divided state of the gold.

Another advantage of the method was that the plant could be confined to a very small space, which was not the case with the ordinary cyanide process.

He did not understand what was meant by stating that "by re-using the amalgam in larger quantities as the base metals were reduced, bullion of about 800 fine was obtained"; did Mr. James mean using larger quantities of the copper-zinc amalgam? He supposed that the object of the copper-zinc amalgam was practically the use of the copper-zinc couple, which seemed to have been used very successfully in South African experiments.

Mr. Picard asked whether pyrites were present in the ore. With regard to the consumption of cyanide in the treatment of the slimes he did not think that, because in South Africa the consumption amounted to less than ½ lb. per ton, it could be assumed that such a low figure would be obtained in the case of the ore now under discussion. He also desired to know how a very weak sodium amalgam would compare with the zinc and copper, as it struck him that by its use the clean-up difficulties would be overcome.

Mr. Sulman said that the author stated that the dry crushed ore was charged into a pan in 2-ton lots with 100 gal. of water so as to form a very thick pulp; that gave a ratio of 4 to 1, which was only 20 per cent. of water—and he knew of no slimes which would form a very thick pulp with that amount of water.

Mr. Claudet thought that the subject of the paper was of great interest to members, for, as the Transactions would show, the Institution had collected a vast mass of information on the cyanide process. While welcoming such papers, it would be well if they were made as complete as possible. In the case before them the author did not give the tailings assay, but he (Mr. Claudet) had calculated it out. For the months of July, August, and September (which were grouped together in the table), as the time when the ore was richest, the tailings assayed 8 dwt. In October, the ore diminished a little in value, and the tailings came down to 4 dwt. In November, when the ore was worth 1 oz. 5 dwt., 7 dwt. was left in the tailings. Later on, when the ore was of still lower grade, the tailings were about 3¼ dwt. It was stated that by re-treating the tailings they get another 6 per cent. out, which on 20 dwt. ore was equal to 1¼ dwt. obtained by re-treatment, the value of the gold was 5*s*., and he desired to know whether there was any profit.

With regard to the zinc and copper amalgam he would like to know definitely what the re-action was. They were told how to make the amalgam, but it was not stated in what form it was put in. He would like to know whether sodium amalgam would, as suggested by Mr. Picard, do the same work.

As to re-using the amalgam, he did not understand it at all, and could not see how a finer quality of bullion could be obtained by

doing so, as was hinted at in the paper. It was a curious fact, however, that if mercury containing gold was treated with a cyanide solution, a certain quantity of the gold could be dissolved out of the mercury.

The costs were stated to be 3*s.* 3*d.* including labour, but that did not appear to be the whole of the costs. What was the cost of the 24 H.P. required? The costs of mining also were not given. He would be glad if Mr. James could obtain the figures.

Dr. Simon said it had been asked if sodium amalgam had been used for precipitation purposes. It had been tried. In the laboratory, on a small scale, it had worked in a perfectly ideal fashion, but as soon as attempted on a large scale complicated reactions took place, which proved it to be an utter failure.

Mr. James said it was unfortunate that Mr. Gilmour could not be present that evening to reply personally to the points raised, but he would no doubt do so in writing.

With regard to the question asked by Dr. Simon and other speakers as to the action of the copper amalgam, he wished to point out that theoretical considerations had been avoided in the paper. The authors were not theoretical men, and had simply related the empirical facts. He took it, however, that the effect of the copper amalgam, if by itself, would be for the copper to go into solution and precipitate the gold on the mercury. Mr. Claudet had referred to the fact that gold was more or less dissolved from the amalgam or mercury, by the action of cyanide, and that the method in question seemed to reverse the process. The copper would do the work by itself without the zinc on dilute solutions, but with a solution of, say, 3 lb. of cyanide per ton of ore treated, the copper alone was not strong enough, and zinc was necessary to improve the action. It might be asked, why not use zinc alone? Zinc amalgam did not appear to be a stable compound, and copper seemed to solidify it and make it more readily workable. As to the increased purity of the bullion resulting from re-use of the amalgam, the action of the cyanide was to dissolve off the zinc, and using the same amalgam over again, the zinc went off more or less completely and then some of the copper, and thus left a fairly good gold amalgam, which was of course less effective in proportion to the diminished amount of zinc and copper present, hence the necessity for the use of larger quantities of such weaker amalgam.

He was much interested in Mr. Sulman's statement in advocating the use of filter presses instead of the method in question. The figures he had seen were certainly much higher than those given by Mr. Sulman. He would like to know where that gentleman

obtained his information. The only costs he had seen from Western Australia were 7*s.* a ton, while Mr. Ashcroft, who had used filter presses on a large scale, stated that the cost was about 7*s.* per ton, although they could be brought down to 5*s.* when working under the best conditions. Even supposing the cost could be brought down to 3*s.* per ton, he did not see how it affected the case, as the cost of the process at the Santa Francisca was under 1*s.* a ton, the remainder being for cyanide.

Mr. Sulman observed that the costs including horse power would come out at something like 5*s.* a ton.

Mr. James said that with regard to Mr. Sulman's remarks about making filter presses that would hold 5 tons per charge, that gentleman seemed to be unaware that there were presses on the market that held 6 tons.

In reply to Mr. Black's question as to drying the ore, he understood that the rainfall at Nicaragua was so limited that they never had to dry the ore, but Mr. Gilmour would probably reply to this question.

With regard to the use of sodium amalgam, it might interest Dr. Simon to know that he also had tried it in the laboratory in the early days of 1888, but the results were not so satisfactory as those obtained by Dr. Simon, and that was one of their reasons for sticking to zinc precipitation.

As to Mr. Picard's question with respect to the costs as compared with South Africa, he thought it was only fair to the authors of the process to state that in looking at the cost of cyanide, which appeared to be so heavy, they should look at the result and take the cost of cyanide consumed per oz. of gold recovered, which was a fair method. If they took the African practice, for every oz. of gold recovered it cost 3*s.* 1*d.* for cyanide, while in Western Australia the returns published by the Associate Gold Mine showed the cost to be just under 3*s.*

Mr. Sulman had also asked how much water was used to make the slimes viscous, and he would obtain the information from the authors.

Mr. Claudet had referred to the high assay of the tailings; he could only suppose that the authors, feeling that they would have a certain amount of criticism to face, had left something for the members to find out themselves. He took it that the later experiments on the sands alone were made because the authors found the assay of the tailings was so high. As to whether it was worth retreating the sands for 5*s.* per ton recovered, it depended on the amount of cyanide consumed. It would be worth while in Africa, where

their treatment costs for sands were down to 2*s.* 6*d.* per ton, and as labour was cheaper at Nicaragua, it might be taken that it was worth while.

With regard to the amount of water, at first only a small quantity was used to get the amalgam well in contact with the ore, but afterwards a further amount was added to settle the amalgam.

The President thought their thanks were due to Mr. James for replying to the questions in the absence of Mr. Gilmour. The paper was a suggestive one, but, as had been pointed out, several matters required explanation.

Pan amalgamation was in most cases more expensive than was indicated by some of the costs given, and the capacity did not seem to be right. For instance, 2-ton lots were charged into a pan for 6 hours—that would be equal to 8 tons a day, and, assuming that they worked four pans, would give a total of 32 tons a day, but as the average of the monthly yield was about 15 tons, it was evident they were not working to the full capacity.

The horse-power for 50 tons a day capacity was probably under estimated, but the process itself seemed to be very like the Pelletan Clerici, and some other processes in which quicksilver was used, with or without electric action. The application of it depended largely on the character of the ores, and the fact that, as pointed out by Mr. Claudet, 50 per cent. extraction was obtained by ordinary pan work, showed that quicksilver played an important part in the operation. The use of quicksilver also introduced the difficulty of the loss of gold, irrespective of the amount of saving by cyanide and the precipitation of the mixed amalgams—such losses always existed in a pan amalgamation process in settling the finely divided quicksilver from the pulp, and the losses varied very much with the character of the ore.

With regard to the water, a charge of 50 per cent. for pan amalgamation was an ordinary one for many ores, and made so thick a pulp that there was no difficulty in dividing and supporting the quicksilver.

CONTRIBUTED REMARKS.

Mr. J. A. Gilmour: In reply to the remarks made by Mr. Sulman, the treatment of the ore as a whole in seven days should refer to San Luis ore, and the 30 days' treatment to the Santa Francisca ore as a whole.

The solution of gold in one hour refers to the gold contained in slimes alone when treated by agitation in presence of cyanide.

The consumption of cyanide per ton, 1¾ lb., is on the ore as a whole, without any separation, when treated in the pans.

Any re-treatment of the sandy portion of tailings is over and above that amount.

Labour costs given do not include white supervision; ordinary wages for this class of labour are from 1*s.* 6*d.* to 2*s.* for 12 hours shift.

The quantity of water used is 100 gal. per charge of 2 tons of dry ore, and this being so clayey makes a very thick pulp. In the settler more water is added, as is the usual custom in pan amalgamation.

In reply to Mr. Claude Black, the ore is sun dried during the summer months, and when the wet season is on plates are employed.

The ore consists of quartz and kaolin; the gold is very fine; silver occurs as black sulphide; pyrites is present in very small amount, and there is a minute quantity of copper compounds.

The remarks as to the increasing the fineness of bullion by re-using the amalgam in the pans may be thus explained. Fresh amalgam being used with each charge in the pan, the amalgam drawn off from the settler still contains the bulk of its copper and zinc, and instead of going to the retort it can be reused in the pans, provided that as the base metals decrease a larger quantity of amalgam is employed.

Instead of drawing off this amalgam when discharging pans into the settlers, the same object would be attained by having concave bottomed pans, and only adding to each charge the actual amount of copper and zinc consumed in the precipitation, then at the end of the month the clean up would recover the amalgam, which contains the gold obtained from all the charges the pan has worked with, only practically the amount of copper and zinc used for *one* charge. The reason that returning the amalgam has not been used to the fullest extent, is owing to the necessity of having honest people to handle it, and we cannot trust the natives to do so without stealing.

With reference to Dr. Simon's remarks as to the action of copper amalgam. In the practical working it was found that zinc amalgam alone presented great physical difficulties in the settlers, as it is liable to crumble into powder and flow off with the water, and also go so far as to blow up into light spongy masses and oxidise, giving off great heat. The addition of copper amalgam prevents all this, and gives a stable amalgam to work with. Copper amalgam alone gives as good precipitation as zinc under certain

restrictions, so it would seem that the effect of the zinc-copper couple is not a necessity.

In reply to Mr. Claudet, who had referred to tailing assays, these have not been given, as it was considered at the mines that the "Bullion Returns" were preferable as the bases of calculation of yield, and where dealing with tailing assays, it often happens that the actual returns are not up to yield got by calculation on that basis.

Cost per horse-power depends so much on the price of fuel and efficiency of the boilers and machinery, that it would appear a matter depending on the locality. In Nicaragua firewood is employed, costing 4*s.* 6*d.* per cord, 4 cords, equal to 1 ton of ordinary coal, say 18*s.* If 3 lbs. of coal per horse-power per hour is allowed, the horse-power per annum will work out about £11 15*s.* 8*d.*

In perusing the discussion on this paper, I would like to point out that it was not the intention to deal with milling or cyaniding of ores, but simply the precipitation of the precious metals from solution in presence of the pulp or magma.

In ordinary practice when dealing with slimes, the cyaniding would most likely be done in a separate vessel, and the agitators used as precipitators only, so that the question of consumption of cyanide would be common to any cyanide process, and would depend on the material under treatment. The loss of mercury is mostly mechanical, and leaving out the cost of this along with cyanide, the other chemicals will not come to more than 3*d.* per ton in ordinary fair working. The labour is simply that of ordinary pan amalgamation, and in large mills can be kept very low.

"Notes on Fouché's Aëro-Condenser."

By Dr. A. L. SIMON, M.Inst.M.M.

WHENEVER power has to be generated by steam, and a water supply of good quality for steam generators is scarce, the adoption of condensing engines is naturally resorted to.

In the generally known condensing engines large quantities of water are used for cooling purposes.

In many mines, and especially in the mines of West Australia, to which these notes particularly refer, water is either very scarce, or it is so heavily charged with salts, that the ordinary condensers

cannot be profitably worked, and air has to be used as a cooling medium.

The first attempts to condense the exhaust steam in the mining centres of West Australia were made with very primitively designed appliances, mostly pipes and drums of corrugated iron. Their efficiency was not satisfactory, and they could only be looked upon as makeshifts, nothing better then being known.

About two years ago, Fouché's aëro-condenser was introduced on to the fields, and has since given such good practical results that the author feels justified in laying these notes before the Institution.

Fouché's aëro-condenser consists of the following principal parts.

A. The ventilator.
B. The condenser proper.
C. The air pump.
D. The oil separator.
E. The driving engine.

The disposition and the working of these different parts is very simple, the only part requiring special description being the condenser proper. It consists of a series of light steel pipes of from ½ in. to ¾ in. diameter. These pipes stand vertically, and connect two square flat-shaped iron recipients, an upper and a lower one. The steam, which is to be condensed from one or more sources, is directed into the upper recipient, it passes through the pipes, which are kept cool by a strong current of air produced by the action of the ventilator, and the condensed water collects in the lower recipient, from where it flows into a well.* The system of pipes is enclosed on two parallel sides with sheet iron, one of the remaining sides is connected with the ventilator, the other is generally open. Should the heated air be made use of for drying purposes the latter side is connected with special drying chambers.

The oil separator is connected with the main pipe leading to the upper recipient. The size of the condenser varies naturally according to the quantity of steam to be condensed.

The following table gives the sizes of the usually built condensers, with the amount of water they are capable of condensing with or without vacuum.

* The air-pump (if any) is connected with the outflow of this lower recipient.

Number of condenser.	Quantity of condensed water obtained per hour, with surrounding air at 86° F.	
	Without vacuum.	With a vacuum of about 15″ mercury.
	gallons.	gallons.
1	66	51
2	77	59·5
3	99	76
4	130	100
5	165	127
6	215	165

Dry air is certainly a bad carrier of heat, on account of its small caloric capacity, and the condensers, therefore, are rather large ones. That is probably the reason why this class of condensers has not been used more frequently.

As regards the economical points the author can hardly give any general figures, the prices of condensed water and salt water being subject to great fluctuations in the various localities. Suffice it to say that at the Norseman Gold Mines, where this Aëro-condenser is connected with one 40-H.P. expansion engine and several engines of minor importance, the daily consumption of condensed water, for one year's average amounts to 50 gallons per day of 24 hours, whereas in one well known mine in Kalgoorlie where a surface condenser is used in connection with a 150-H.P. engine, the amount of water lost for cooling purposes amounts to 18,000 gallons, or 144*s*. per day. When such amounts are lost, the capital expenditure which may be required for the installation of a thorough air condenser, in which the repairs and the up-keep are almost nil, is well justified.

There being no discussion on the above,

The President said that there were two other papers before them, but owing to the lateness of the hour they would be carried forward to the next meeting.

"The Tin Mines of Bolivia."

Communicated by WALTER McDERMOTT (*President*).

THE author of this paper, Mr. Chas. S. Pasley, has spent eight years in Bolivia, during most of which time he has been actively engaged in working tin and silver mines. He also made a tour of inspection, visiting nearly all the tin mines in the country, so that the following remarks are made from personal knowledge.

Mr. Pasley was manager of two of the tin mines mentioned in this paper, those of Colquiri and Totoral. The specimens of ore are chiefly from the latter mine, and will give a fair idea of the richest ores obtained from these mines, which are believed to be by far the richest in existence.

GEOGRAPHICAL.

The tin-producing districts of Bolivia are situated on the Eastern Cordillera of the Andes, between latitude 21° S. and latitude 16° S., extending north and south over a distance of 300 miles, appearing in distinct groups. They are all to the eastward of the great Bolivian plateau, in which are the lakes of Titicaca and Poopo.

No tin mines are known to exist in the Western or Coast Cordillera, and the author has not heard of any tin being discovered in the neighbouring countries of Peru and Chile.

The Eastern Cordillera consists of numerous ranges running nearly parallel to each other, and further east of spurs thrown out at right angles, one of these reaching nearly to Brazil, dividing the waters that flow to the Amazon and Parana. The highest range is that nearest to the plateau. The lowest passes of this range are over 14,000 ft., and includes some of the highest mountains in South America, such as Sorata or Illampu, Huayna Potosi, Illimani, Tres Cruces, and others of great altitude. The more eastern ranges are lower, and broken up by deep valleys, where numerous small rivers pass to the tributaries of the two great rivers.

The plateau extends for over 400 miles north and south, with an average width of about 80 miles. At one time it was all one huge lake, as the old water marks of lime are visible for large distances, running along some of the hill sides like a white band.

GEOLOGICAL.

The Andes in this district are of the Silurian series, and include the rocks of both the Upper and Lower Silurian formation, abutting to the westward on the old lake bed. This formation is of great thickness in the valley of the Arque, at the foot of the mountain of Berenguela, a height of 6,000 ft. above sea level. Nearly the whole stratification can be seen plainly up to a height of 14,000 ft., and it is the same up to the summit of Tres Cruces, at the height of 20,000 ft.*

This range has been disturbed by eruptions of porphyritic rock, which shows plainly in isolated groups of hills, such as Oruro, Sicasica (Garce Mendoza), and others, which formed islands in the great lake; these porphyritic rocks are the chief feature in this range. Prior to these eruptions of porphyry has been the intrusion of the granite, which is abundant in the eastern ranges, but is more seldom seen in the main range.†

These eastern ranges consist of slate or layer upon layer of greywacke, shales, and thick beds of sandstone; they are generally auriferous. Further to the south the eruptive rock is trachyte, and near to Chile and the Argentine Republic the rocks are gneissic, or schists broken up by granitic intrusions.

On the western side of the lake bed the formation is distinct with rocks of the oolitic series, with a belt of Permian formation, in which are situated the copper veins of Coroccoro, San Bartolo, &c.

There are scarcely any signs of volcanic action east of the old lake bed, except at La Paz, and earthquakes are unknown.

Nearly all the tin lodes occur in the porphyry, or altered andesite, but in a few places the lodes run through slate and trachyte. The veins in the porphyry were generally the richest and of better quality, the others being mixed with antimony, iron and copper pyrites, zinc blende, and sometimes with bismuth and wolfram.

* The cleavage in general is from north to south with the dip to the eastward, most of the lodes cutting the stratification at right angles. This appears to be the same in the silver mines.

† The author has never seen any tin lodes in the granite, but has heard of stream tin being found in the Tipuani and Cajones rivers by gold prospectors. In the mountains where these streams have their sources granite is plentiful.

The tin usually has traces of silver, and the lodes frequently have a capping of iron.

MINES.

The tin-producing districts may be divided into four: Oruro in the centre, La Paz in the north, Chorolque in the south, and Potosi further to the eastward.

Oruro is by far the most important of these districts, the amount of tin sent from Oruro being above two-thirds of the total production of Bolivia. The following is a brief description of the principal mines of the country, commencing with the hill of Huanuni—the most important mines of the Oruro district.

Huanuni Hill.—The mountain of Huanuni has an imposing appearance, rising over 2,500 ft. above the river which flows at its base, and is about 15,500 ft. above sea level.

The principal lodes run nearly east and west, dipping to the south, cutting the strata at right angles in one part of the hill, and at an angle on the opposite side. The stratification is very irregular, and there are a number of faults. Four of the principal lodes run nearly parallel, and have a dip of about 50° to the south. These lodes run across to another hill of slate, their character changing with the different rock, being mixed with arsenical pyrites. There are three or four other cross lodes, but these are not nearly so rich—these cross lodes seem to be of later formation.

There are several proprietors on this hill, the chief owner being the Huanuni Company and a Bolivian firm. There are also about half a dozen others.

The lode worked by the Huanuni Company, named the Catarecagua, varies in width from 2 ft. to 8 ft. The ore is fairly continuous throughout, but has rich shoots. The Spaniards, who were former owners, worked down on one shoot to a depth of 1,000 ft. vertical. This part is now cut by a tunnel and worked by more modern methods. The old working is used for ventilation; formerly the ore was partly sorted at the bottom of the mine and then carried out on men's backs—a man could only make two journeys in a day, carrying 75 lb. at a time. There are signs of galleries in several places, but these were all blocked up with the dirt carried up from below.*

The tinstone is generally richer on the foot wall, but the tin is

* The author was informed by the manager of this mine that on a conservative estimate there remained over 5,000 tons of tin stone in sight at over 25 per cent., but he was unable to work it until he had got the lower part of the mine in order. The dirt left in these old workings contained 20 per cent. of tin oxide.

fairly good throughout. Both the rock and the vein matter are very hard. The sorted ore from the lode, as a rule, contained about 50 per cent. of oxide, and the poorer stuff about 20 per cent. The ore picked over by women, who generally throw away any stuff containing less than 10 per cent., is all put on one side to be worked at some future time, as there is sufficient ore of the richer quality to keep the floors going. This mine produced about 100 tons of concentrates per month.

This company has a 10-stamp mill and several jigs and round buddles, the floors being worked on a modified Cornish system which was very successful. The tailings from the floors contained about 5 per cent. of oxide. These were all saved to be worked over again.

The author was not permitted to examine the workings of the native firm, as there were rumours that they had passed the limits of their claim.

The ores that were being sorted were extremely rich; some of the ore gave 65 per cent. of tin without concentration, and the dumps ran about 15 per cent. of oxide. They had struck a very rich pocket in soft ground, and had taken out 2,000 tons of concentrates.

Their work was all done on the old Spanish system—all the ore and dirt being carried on men's backs, and the floors were managed in the same fashion. The ore was washed by women in a narrow trough about 12 to 18 in. wide. The bottom of the troughs are made of peat with a curve so as to gradually lessen the force of the stream of water. The ore is moved up against the water by a thin piece of wood a little shorter than the width of the trough until the dirt has been carried off.

This process is exceedingly wasteful, nearly all the slime tin being lost. The ore was crushed by a Chile mill, worked by mules.*

Several of the other mines appeared to be important, but were badly worked. The total production of the mines in this hill amounted to about 250 tons of concentrates of 65 per cent. per month. Some of the lodes in this hill had a large quantity of pyrites, but these were not worked.

Close to Huanuni, at the base of the hill, are several veneros or deposits of stream tin; some of these were worked on a small scale on tribute; the quality was very good, but the quantity produced very small.†

* A new tin dressing plant on the same plan as the Huanuni Company has recently been ordered by this firm.

† The extent of the "veneros" was very large. These had been prospected by the natives in various places, and the best patches worked. There was

Morococala.—Ten miles north of Huanuni are the mines of Morococala, Vilacollo and Negro Pabellon, on hills of the same names, At Morococala are three mines with different owners. The principal one is worked by a shaft sunk on the lode, which is 12 to 15 ft. wide in places, and carries very rich and pure oxides. The rock is hard, and the work had been abandoned for want of capital and legal trouble. There was water at the bottom of the shaft, the depth of which was about 150 ft. The ores and dirt are taken out by means of a mule whim.

The other two mines were working on a small scale—one worked by a Chilian miner had a small shaft with a 2-H.P. engine for drawing out the water and dirt. The third mine was worked by an adit. All of these mines were similar, with lodes east and west, nearly vertical. The ore taken out averaged about 25 per cent. of oxide. The floors were all on the native system, the crushing being done by a Chile mill.

Vilacollo, a short distance from Morococala, was abandoned, and most of the workings had caved in owing to the softness of the ground and want of timbering. Large quantities of ore had been taken out of this mine in former years. The quality of the ore was bad, the percentage of iron being very large. A caretaker, who had worked in the mine, said that it was not a lode, but only a shoot. There was no trace of walls; the surrounding rock was sandstone of a deep red colour, impregnated with iron. All the dirt taken out of the mine contained tinstone mixed with iron oxide and pyrites. A shaft of 250 ft. had been sunk at the bottom; there were no traces of tin, the metal containing a little silver with arsenical pyrites. The floors were three miles from the mine, there being no available water nearer. They were of the native type, with a large Chile mill for crushing.*

Negro Pabellon, a mine owned by a German firm, and worked on tribute by a Chilian miner, was of quite a different type to the others mentioned, the lode runing north and south, and vertical, and in the same direction as the cleavage. The dip of the strata was about 45° E. The country rock was slate, and the lode contained the

plenty of water, but very little fall could be obtained. Some of the patches had too much iron, but in many places the stream tin was in large rounded pieces weighing a couple of pounds, and nearly pure oxide. No system was observed in working the stream tin—each man working where he pleased on tribute, and generally stealing about half the tin that he obtained.

* The owners of this mine had erected excellent houses for their staff and workpeople. These had been completed shortly before they had shut down the mine. At this time they had no capital left, and had never done any development work except the shaft.

same rock and tinstone mixed with iron pyrites. The tin was very rich and easily worked, the ore averaging 20 per cent. from wall to wall. The lode seemed very regular, about 3 ft. wide, crossed by several small veins; rich pockets were encountered wherever the lodes crossed. The mine had not been worked to a greater depth than 100 ft. The tinstone was soft and easily washed and crushed; the concentrates gave a very high percentage of tin, up to 70 per cent.

The floors were of very small dimensions, as the water was very scarce. Three hand jigs and various troughs for hand washing. The crushing was done by a Quimbalete, or rocking stone. A short distance from the mine were old floors with an iron overshot wheel for driving a 4-stamp battery, but the water was not sufficient to turn the wheel round.*

The mines of Morococala and Negro Pabellon are about 15,000 ft. above sea level. In each of the watercourses between Huanuni and Negro Pabellon were found one or two people washing stream tin. The principal venero was at Japo. This stream tin was in patches and of good quality, but there was very little water in the streams. The natives who work in this way are quite contented if they collect and sell 1 cwt. per week per man.

Very often these stream tin washers will collect about 10 to 12 cwt. This they sell a day or two before a big feast day, in order to have a long spell of drunkenness.

Colquiri.—50 miles to the north-east of Oruro are the mines of Colquiri. They are situated in the province of Inquisivi, which officially belongs to La Paz, but it is over 100 miles from that town. The principal lode, which is of greater width, was worked by the Spaniards for silver chlorides. The deepest working that was discovered by the author was 175 ft. from the surface; this reached the water level. This lode can be traced for 3 miles on the surface, having immense "desmontes," *i.e.*, dumps all the way along.

There is a tunnel about 1,000 ft. long cutting the lode just above water level, and in another part an adit nearly as long. The vein matter is 80 ft. wide in the tunnel. The walls are well marked and vertical. The lode runs N. 80° E. and S. 80° W. The tin on the south wall had been worked on either side for 150 ft., and had been stoped out up to where it was covered by the silver chlorides. The tin had apparently been in pockets, as the width of the work-

* As a specimen of the mechanical skill of the people who worked this mine, they told me the reason that the wheel would not turn was because the air in the buckets would not let the water act as it should have done, and they intended to bore holes in the buckets so that the water might drive the air through.

ings varied from 2 ft. to 10 ft. By the books, the people who had worked it had sent away over 500 tons of concentrates. The author found specimens that were mixed with pyrites that gave 75 per cent. of oxide. Beyond this streak of tin, the vein matter consisted of arsenical pyrites, magnetic iron, copper pyrites, heavy spar and iron oxide, with narrow veins of galena about half an inch wide, dipping across the other vein matter to the north. These galenas contained about 100 oz. of silver to the ton. Beyond these, next to the northern wall of the lode, was a wide streak of grey clay mixed with iron pyrites in large lumps, with tin and fluor-spar disseminated throughout the clay. Water was filtering through this streak, which averaged 8 ft. wide. It was expensive to work and had been abandoned, as the stone archwork was too expensive to build. The tin stuff averaged 4 per cent. A shaft had been sunk in the middle of the lode for 75 ft. This was full of water, which could not be handled without machinery. The water that ran out of the tunnel was used for washing on the floors. An analysis of this gave 1½ per cent. sulphuric acid and 1 per cent. of copper. The ores were first concentrated to about 40 per cent., and then roasted in a reverberatory furnace and re-concentrated.

Twenty miles north of Colquiri is the Tres Cruces, a large group of mountains 20,000 ft. high.*

The only tin lode worked was Sayaquiri. The lode was about 25 ft. wide; the tinstone was mixed with iron pyrites, and also contained wolfram; wherever any sinking on the lode had been done the tinstone got worse in quality. This lode ran E. and W., and was nearly vertical; in other lodes the tin was remarkably pure, in brown coloured crystals, the concentrates giving 70 per cent. of tin, but these veins were at a great height, far from water, and snow-covered half the year.

Here was found a large block of nearly pure oxide, about 2 ft. thick, among the detritus, at 18,000 ft. The lodes contained tin mixed with antimonial silver, running at nearly right angles to veins of gold-bearing quartz, but one could never see the actual place of crossing or where they had been cut off; also this part was too far from the railway, 85 miles by road, making the freights too high.

The mines of Oruro are well known as silver mines; most of the lodes have tin on the surface, which sometimes penetrates to a considerable depth, mixed with the silver ores. All the work is carried on for silver, no account being taken of the tin until the silver has been extracted by amalgamation (with the exception of a

* All the tin lodes in these hills were at the southern end; in the eastern side all the veins were silver and galena, and to the north auriferous quartz.

few contractors who work the tin stuff found on the dumps and close to the surface).

There is no water at Oruro available for floors; the ores are carted to the amalgamation establishments about 15 miles off. There are two of these, "Machacamarea" and "Obrajes."

The tailings of the silver ores are washed for tin; about 50 tons of concentrates are obtained monthly. The tin is not of good quality and is very variable in quantity. It is nearly all worked on tribute. Frue vanners were used for concentrating.

At Oruro there are several large deposits of slags left there by Spaniards who smelted the Oruro ores for silver; these slags contained 30 per cent. of tin and 20 per cent. of lead and some silver, A quantity was shipped to the coast to be re-smelted, but did not pay.

Close to the establishment of Machacamarca, at an abandoned silver amalgamation works, a Frenchman had erected a water jacket furnace for tin smelting, but no particulars are forthcoming as to whether it has been successful or not.

At the mines of Challa-Apacheta, 12 miles south of Huanuni, which were worked for some time by a Cornishman, there is another curious instance of a very wide lode, which is about 120 ft. wide, with the tin in patches, the vein matter consisting of clay and iron oxide, the tin being principally slimes. All the tinstone was mixed with arsenical pyrites and iron oxide; the vein matter was very soft and could be worked with picks. The tin averaged about 2 to 3 per cent., but was easily worked. The slimes were of very good quality. Whenever a good sized hole had been made, a cave in would take place, and the dirt wheeled out to the floors until no more dirt would fall. There were workings on one side reaching a depth of about 100 ft. The ore here was very dirty and of bad quality; the lode appeared to be changing into silver pyrites at this depth.

The floors were worked by the Cornish system, with square buddles and kieves. There was a scarcity of water in the dry season.

The Mines of Abicaya.—About three miles to the south-east of this mine is the hill of Challa Grande, with the mines of Abicaya, Totoral, and Chuncho.

The veins in the two first named run east and west nearly parallel, varying in their dip from 60° to 40° to the north.

The lodes are all in very hard porphyry, the tinstone being very pure on the foot walls, but sometimes on the hanging walls, with shoots of ore occasionally from side to side, the veins varying from

1 ft. to 5 and 6 ft. wide; all the vein matter was very hard, and had a lot of quartz with it. The tinstone was sometimes in fair sized black crystals, but generally in compact masses. The tinstone taken from the walls averaged 40 per cent., and the rest of the vein 20 per cent.; the lodes were all well formed and very regular. These mines are situated on a spur of the hill, one on each side, but Abicaya being 400 ft. lower than Totoral, Chuncho is still higher and near the centre of the hill. The lodes here run north-east and south-west, and are vertical, crossing the lodes of Abicaya and Totoral. The tinstone in these lodes was softer, but more mixed with pyrites and iron oxide. On the opposite side of the hill there are several powerful silver lodes, these running east and west, and nearly in line with the Abicaya lodes, but the dip was nearly vertical.

The floors of Abicaya consisted of a Gruson ball mill, a three-compartment jigger, and a Frue vanner, which seemed to suit the class of metal fairly well. These were driven by a turbine; this could only be run half time during the dry season. The floors of Totoral consisted of Cornish jigs and square buddles, the ore being crushed with rocking stones shod with iron.

Further in the interior, about 30 miles east of Abicaya, are the mines of Uncia and Llallagua, both on the hill of Uncia. The road of Llallagua runs east and west, and nearly vertical. The mine was badly worked; the tinstone was rich and of good quality, occasionally mixed with pyrites, and often occurring in masses of large crystals. The lode contained a large amount of oxide of iron; the ground was soft and easily worked; the floors consisted of square buddles, with a rocking stone for a crusher.

At Uncia the mines were worked on tribute; the lodes were wide and well formed; one of them, a cross lode, had an adit about 300 ft. in length; the tin was mixed with silver and copper pyrites. All of these mines belonged to natives, who were hopelessly in debt to a German firm. There were some very good veneros, and the tributors brought in a fair quantity of stream tin of very good quality.

Further in the interior, about 70 miles east of Oruro, are the mines of Berenguela in the mountain of the same name. These mines are the furthest east of the Oruro district, and at present all work has been suspended owing to lawsuits. The lodes, of which there are several, are wide, apparently running north-east and south-west, with a dip to the north. The mouths of only two of the mines were visited by the author, who had to judge of the others by the

lines of "desmontes." The tin stuff is nearly all in slimes, and ran about 6 to 7 per cent. on an average, and of very good quality. The hill is exceedingly steep; the mines are nearly 13,000 ft., whilst the valley at the base of the mountain is 7,000 ft. above sea level. On one side of the hill there were masses of purple porphyry, and lower down several spurs of the mountains appeared to be of red sandstone. Trees grew wherever they could get sufficient hold, and water, both for washing and power, was more than enough. These mines belong to the department of Cochabamba, their outlet being by Oruro.

La Paz.—The mines in the La Paz district are not of great importance; one of them, Chacaltaya, had a wide but very uncertain lode, and had floors consisting of a Huntingdon mill, jigs, Frue vanners, &c., driven by a Pelton wheel; but the tin production was insignificant. The most important mine was that of Milluni. The work was left almost entirely to tributors, who had been allowed to ruin the mine. There were several small workings on the hill of Huayna Potosi; these were at the snow level, and were not visited by the author. The whole tin production from the La Paz district is slightly over 300 tons of concentrates a year.

Chorolque.—The mines of this district are the most southern. They are situated on the hill of Chorolque, a peak over 18,000 ft. The mines are 16,000 ft. above sea level. There are several powerful lodes running east and west, and nearly vertical. The principal lode is worked for tin and also for bismuth, the tin being in one part of the lode and the bismuth in another.

These mines were worked by German engineers; the floors consisted of a ball mill, jiggers, and Frue vanners; also, there were two furnaces for smelting tin and two for bismuth. The bar tin was of good quality. The same firm that owned these mines were also owners of some other tin lodes in the hill of Chocaya, a few miles from Chorolque, but these were not worked.*

Potosi.—There are not any mines in the hill of Potosi that are especially worked for tin. As at Oruro, the silver lodes are mixed with tin near the surface. Most of the tin stuff is taken from the dumps of the old workings by tributors, who sell it to a smelter; these smelting works use water-jacket furnaces; the bar is of very bad quality, as the metal is very much mixed with antimony and other metals.

The fuel used is charcoal and llama dung. The smelting pays

* The owner of these mines has informed me that he has recently ordered a complete concentrating plant from Germany, which has proved very satisfactory.

well, as the railway is 150 miles off; a good quantity of bar tin is sent from here.

Ten leagues from Potosi are the mines of Porco. These mines are not worked; they were abandoned, as the ore was too poor to work. The tinstone ran about 7 per cent. on an average; there were a number of lodes, and the floors were worked by water power.

To the north of Potosi are the mines of Maragua and the veneros of Ocuri. The Maraguas mines were worked by fits and starts; they were not paying. The lodes, of which there were a great number, were narrow and close together. The tinstone was rich. No proper work had been done to develop these lodes, as the work was done by tributors.

The veneros of Ocuri were worked by two Cornishmen, and had given excellent results. The tin was plentiful and of very good quality. There was plenty of water in the river, and the work was carried on systematically. These veneros were a long distance from the railroad—over 90 miles.

Many lodes exist in all of these districts which cannot be worked, as they are claimed by people who never intend to work them. Anyone can hold a mine on payment of a small fee and a yearly fee of five Bolivian dollars—less than 10*s*.—a year per hectare. These people do not pay the yearly fee after the first year; but if anyone else claims the mine the result is a lawsuit which may last for years.

The progress made of late years in the tin mining industry is due chiefly to the completion of the Antofagasta and Bolivia Railway, which was opened to Oruro in 1892. This has lessened the freights and thereby enabled many people to start work; formerly all the traffic was carried on by mules, donkeys, and llamas from Oruro to the port of Arica in Peru. Before the opening of the line the cost per cwt. used to be:—

By mules in 5 to 7 days	about	10*s*.
By donkeys in 12 days	„	8*s*.
By llamas in 30 days.....	„	5*s*.

It is now about 2*s*. 6*d*. per cwt. in three days by rail. The difference with regard to machinery is still greater, as formerly all parts had to be made for mule transport.

The smelting of the ores up to the present is not on any large scale. The ores smelted at Quechisla from the Chorolque mines would probably pay as well if sent away as concentrates, and the ores smelted in Potosi are those of very bad quality, the concentrates being so dirty that no merchant would purchase them, except

at a very low figure; for this reason the smelters buy them at a very low rate, and make money by saving on freights. The railway company charge double freights for all goods sent from the coast. The cost of coal, chiefly Australian, is about £8 per ton delivered at Oruro—a prohibitive price for smelters. The object of the railway company is to secure as much freight down to the coast as possible.

Workmen are easily obtained, though they are not as plentiful as formerly, as many have been killed off by the cheap alcohol, which has almost become the national drink. The men and women will work hard, and also steal metal, in order to obtain this. Most of the mining is done by Cholos, *i.e.*, half breeds. Indians can be obtained for outside work, building houses, making ditches, &c. They are not as highly paid as the Cholos, and are much less trouble to look after, though they are rapidly becoming as expert at thieving. These robberies are very difficult to avoid. There are always a lot of ore purchasers in the small villages, who give alcohol in advance to the workmen; this is considered a debt of honour, to be paid by robbing ore as soon as possible.

Often these receivers are the local authorities in the villages. The merchants are not much better, as they buy any ore that is brought to them without making inquiries, or trying to, often buying ores that have been stolen from their own mines. The number of workmen and women employed in the various tin mines and floors does not exceed 3,000.

These districts are all at a very high altitude, and are very healthy. As a rule Europeans soon become accustomed to the rarefied atmosphere, and live perfectly well at 16,000 ft. From May to August the cold is intense, the thermometer often falling below zero (Fahr.). The differences of temperature are very trying at times. The thermometer has shown 70° in the sun, and 10° of frost in the shade, as there is hardly any radiation at these heights. A very high percentage of the deaths occur directly or indirectly from alcoholism, both with natives and Europeans. As the author has not been able to obtain the statistics of 1897, an approximate table of the tin exploited at the various mines is appended.

The total production of tin in Bolivia, according to the yearly statement of the Minister of Finance, taken from the amount of export duty paid, represents 5,910 tons of bar tin; but in these figures the concentrates are not separated from the bar tin, so that the exact figures cannot be given.

In the year 1893 the exports amounted to 2,000 tons, so that in five years the production of tin has been trebled. These figures

were received after the author had made an approximate table of the amounts produced at the various mines.

The export duty charged is:

50 cents per 100 lb. of bar tin ..... 9*d.* nearly.
35 „ „ concentrates 6½*d.* „

The author is indebted to Señor Avelino Aramayo, the Bolivian Minister in London, for these figures.

Location.	Name of mine.	No. of lodes working.	Direction.	Dip.	Rock.
Huanuni Hill....	Catarecagua ...	1	E. & W.	50° S.	Porphyry.
„ „	Barreno.....	1	N.75°E.	60 S.	„
„ „	San José	1	N.80E.	45 S.	„
„ „	Dejada.....	1	E. & W.	50 S.	„
(Penny Bros.) ...	Vetilla.....	1	E. & W.	70 S.	„
„ „ ...	„	1	N.70E.	..	„
„ „ ...	Various	..	..	..	„
Negro Pabellon ..	Negro Pabellon	3	N. & S. (2 cross lodes)	vertical	Slate.
Morococala.....	..	3	E. & W.	„	Porphyry.
Vilacollo	Vilacollo	1	..	..	Sandstone and slate.
Colquiri	Socabon Verde.	1	N.80E.	vertical	Slate
Tres Cruces	Sayaquiri.....	1	E. & W.	„	Porphyry.
Oruro	Various	..	..	..	„
Challa Apacheta..	Challa Apacheta	1	N.45E.	vertical	Slate.
Challa Grande ...	Abicaya	4	N.80E.	50 N.	Porphyry.
„ „ ...	Totoral	4	N.80E.	50 N.	„
„ „ ...	Chuncho	3	N.E.	vertical	„
Uncia.....	Uncia	2	N.80E.	„	„
„	Llallagua.....	1	E. & W.	„	„ and slate.
Berenguela.....	Various	..	..	..	Porphyry.
La Paz	Chacaltaya	1	N.50E.	..	„
„	Milluni				
Chorolque	Santa Barbara..	1	E. & W.	vertical	„
Potosi	Various	..	..	..	„

Location.	Name of mine.	Height above sea level.	Monthly production in tons of Barilla.
Huanuni Hill.....	Catarecagua	14,500	100
" "	Barreno	15,000	"
" "	San José	"	"
" "	Dejada	"	"
" "	Vetilla.....	"	25
" "	Various.....	14,500	10
" "	Vetilla.....	"	15
Morococala.....	Various.....	15,500	30
Negro Pabellon	Negro Pabellon	15,000	30
Vilacollo	Vilacollo	14,000	..
Colquiri	Socaben Verde.....	"	10
Tres Cruces	Sayaquiri	15,000	10
Oruro	Various.....	13,000	50
Challa Apacheta	Challa Apacheta ...	14,000	5
Challa Grande	Abicaya.....	15,000	35
" "	Totoral	15,500	35
" "	Chuncho.....	"	..
Uncia	Uncia	14,000	10
"	Llallagua	13,000	35
Berenguela.....	Various	14,000	..
La Paz	Chacaltaya.....	15,000	25
"	Milluni.....	14,000	25
Chorolque	Santa Barbara	16,000	35
Potosi	Various.....	14,500	20

About 70 tons of bar tin is sent monthly from Chorolque and Potosi—600, or 7,200 yearly.

In the figures of production other small mines contribute; these figures are difficult to obtain correctly as the owners are given to exaggerate the output of their mines, and the total of 7,200 tons of barilla yearly of 65 per cent. of tin, equal to, say (with loss on smelting), 4,650 of tin, counting the bar tin as 1,000 tons yearly, gives a total of 5,650 tons.

Mine.	Sn.	Fe.	Gangue, &c.	Remarks.
Catarecagua, Huanuni ..	37·05	6·75	56·20	Average value.
Barreno.....	79·00	4·25	16·75	Picked ore.
Morococala.....	68·00	7·15	24·85	Traces of silver.
Socabon Verde, Colquiri	4·30	15·20	80·00	" "
" "	72·00	7·50	20·50	Picked ore.
Oruro, Salinas.....	17·25	12·25	70·50	Traces of silver.
Totoral, Challa Grande .	84·50	3·25	12·35	Picked ore.
Uncia, Llallagua	46·00	14·00	40·00	Traces of silver, some antimony.

"Minerals found in the Silver Lodes of Tatasi and Portugalete, Bolivia."

By MALCOLM ROBERTS, Assoc. Inst.M.M.

THESE lodes, running in a general easterly and westerly direction, with a variable dip, are situated at an altitude of about 12,000 ft. above sea-level, in the province of Sud-Chichas, department of Potosi. They traverse a volcanic tuff (containing quartz, decomposed felspar, and a little decomposed mica), which on the surface is of a few leagues in extent, and is surrounded by schists and sedimentary deposits. The following minerals have been determined as occurring in these lodes.

Native Silver.—Local name "pasa-mano."—Occurs from time to time in a wiry, interlaced form. When found it curiously enough is considered a bad omen, as experience appears to have taught that it is a sure sign of the lode becoming poor.

Pyrargyrite.—Local name "rosicler."—Good crystals are rare. Sometimes it is found in considerable abundance in rich bunches of ore.

Pyrostilpnite (*Fireblende*).—Occurs very rarely in small, flattened acicular crystals.

Miargyrite.—A single massive specimen has been determined.

Tetrahedrite (*Fahlerz*).—Local name "cochizo."—This mineral forms one of the most abundant silver-bearing ores, and contains as high as 17½ per cent. of silver. Crystals are not common, the meneral being generally found massive, and in small patches and specks in the pyrites.

Jamesonite.—Local name "acerado."—Under this head are classed minerals composed of antimony, lead, and sulphur. These are argentiferous, and give a good percentage of the silver ore exploited. Occurring massive, mixed up with pyrites, blende, galena, &c., they are always of a steely appearance, but differ a good deal in structure; thus there is a fibrous variety ("acerado hebroso"), a slaty variety ("acerado lajizo"), and a whitish variety ("acerado blanco"), besides others which in the fracture more or less resemble steel. Their assay value for silver is never so high as that of fahlerz, and

seldom passes 1·5 per cent., being generally about 0·5 per cent. when pure.

Plumosite.—Local name "paja-quemada."—A capillary variety of jamesonite, resembling fine, burnt straw. It occurs in vugs, and is argentiferous, giving as much as 0·6 per cent. silver, although generally much poorer.

Bournonite.—Is found somewhat rarely in cruciform crystallisations, and contains some silver.

Stibnite.—Although never abundant, it is present more frequently in the deeper levels.

Galena.—Local name "liga."—In some lodes this is the predominant mineral, and is found, without exception, in every lode in greater or less quantity. It occurs in cubes and cubo-octahedra, massive and granular. When coarse and in big crystals (called "carne de vaca") it is relatively a low grade silver ore, and the finer variety ("chini-liga") is always richer. The assay value varies, but when no other silver-bearing mineral can be distinguished, it does not give more than 0·5 per cent. of silver, and, as a rule, gives under 0·3 per cent. Galena has been, and sometimes is now, one of the chief silver producing ores, and it may be of interest to note that, even when almost pure, it is treated by amalgamation in Bolivian tinas, with a loss of about 18 per cent. silver and 2 oz. of mercury per marco (7·394 oz.) of silver extracted.

Pyrites.—Local name "bronce."—Iron pyrites is most abundant, being in this that the silver-bearing minerals, fahlerz, jamesonite, &c., are most frequently embedded. The crystallisations are octahedra as a rule, cubes being less frequent, and often the crystals have a thin coating of a yellow to black colour.

Copper pyrites ("cori-bronce") is not common, but, when found, is taken as a sign of rich silver mineral being met with.

Mispickel.—Local name "bronce-blanco."—Also is not common, and is considered as indicating poor ore.

Blende.—Local name "chumbe."—This mineral is nearly always present in the lodes, crystallised and massive, varying in colour from yellowish-brown to black, the latter colour being the most congenial to the silver miner. It has a low silver-assay value, even when fahlerz, &c., cannot be noted embedded in it.

Chalybite.—Local name "cachi."—Occurs in crystals, needles, &c., in the vugs. Sometimes augurs well for the presence of rich fahlerz.

Limonite is found iridescent in oxidised ores.

Quartz.—Local name "quijo."—Occurs in crystals and massive, and often has specks of rich fahlerz.

Anglesite.—Has been found crystallised in gossan.

*Augelite.**—Crystals of this rare aluminium phosphate have been distinguished in two specimens.

Kaolin.—Local name "jaboncillo."—Is found in small deposits, coating vugs and joint planes.

Calcite.—Occurs somewhat rarely.

Gypsum.—Rarely, as small crystals.

Cervantite.—Is found now and again in small, yellow crystals, and, perhaps, some yellow coatings on pyrites are of this mineral.

Valentinite.—Has been seen in several specimens, all of which have contained rich silver ore.

Mr. L. J. Spencer, M.A., F.G.S., of the Mineralogical Department of the British Museum, has examined samples of these ores, and the author is indebted to him for dispersing several doubts, and for noting the rarer minerals mentioned above.

* *Mineralogical Magazine*, 1898, vol. xii, p. 1.

FOURTH ORDINARY MEETING, 18th January, 1899.

Mr. WALTER McDERMOTT (*President*) in the Chair.

The minutes of the previous meeting were read and confirmed.

The President announced that these two papers were left over from the last meeting and were now open for discussion. With regard to that communicated by himself, Mr. Pasley, who furnished the notes, was present, and would be glad to give any further information desired. He thought that the subject of tin mining was at the present time of importance, owing to the improved prospects of some of the Cornish mines, and the high price which the metal commanded. The figures given by Mr. Pasley, as compared with English tin mining, indicated exceedingly rich ore. He presumed that enough developments had not taken place to enable one to state that any large quantities of such ore were in sight, but the ore that had been worked gave an exceedingly high return, although the processes in use were extremely crude. Many of the conditions of working were unfavourable, especially in the matter of fuel and transport. Although the district was now in railway communication with the coast, the freight rates were very high, and he understood that fuel was very expensive. With regard to the latter item, however, the amount of power necessary for concentration work was very small, and therefore the question of fuel was not of great importance. He believed that no attempt had been made to smelt the concentrates on the ground, but that they were shipped to England, and sold to the smelters in this country.

Mr. J. H. Collins thought they were much indebted to Mr. Pasley for sending the information, and to the President for bringing it before them. The paper contained information which those engaged in tin mining were not often able to obtain. He observed that the geological conditions were, in a general way, much the same as obtained in the west of England, *i.e.*, there was a mass of palæozoic stratified rocks which had been pierced first of all by granites, and secondly by porphries, and it appeared that the second intrusion

had brought in the greater part of the tin. No mineralogical particulars were given in the paper, and he noticed that there was no mention of the mineral schorl. He would be glad to hear whether any schorl was found in the country, as that was the most characteristic mineral in association with the tin of Cornwall; he had never seen any oxide of tin in any part of the west of England without usually finding schorl within a quarter of an inch of it. It appeared that there was a very close association of silver, which was not unknown in Cornwall, although it was very rare. It had been found in the North Dolcoath, but was not as a rule characteristic of the tin districts of Cornwall.

The Bolivian deposits were undoubtedly very rich, and he considered that Cornwall had more reason to dread in the future the production of tin from that country than from any other part of the world where tin ores were known at present.

He hoped that the paper would be supplemented by further information, more particularly with regard to the mineralogical characteristics of the ores, that cross sections would be furnished, showing the manner in which the lodes occurred. Sections of the veins themselves were also desirable, in order that they might see whether they were or were not composed (as many of the lodes in Cornwall were) of brecciated material.

Mr. Claudet asked if any complete analysis had been made of the ore in question.

Mr. L. J. Pasley observed that, as Mr. Collins had pointed out, schorl was always found near the oxide of tin. With regard to Mr. Claudet's remarks, he had no analysis of the ore, although a good many specimens were to be seen in the Jermyn Street Museum. He thought, however, he might be able to obtain some further specimens, and have an analysis made for a future paper.

The President said that Mr. Collins had referred to the occurrence of silver; in certain districts of Bolivia this was so marked as to make the working for silver as important, and sometimes more important than working for tin. English companies had had experience in that direction, working by the Francke process of amalgamation, and subsequently concentrating the tailings for the extraction of the oxide of tin. The ores were very base in character, and offered considerable difficulty in connection with economical working under present conditions, but he thought that they would in future constitute an important subject for investigation, and, doubtless, for the investment of capital as well.

"The Deep Alluvial Leads of Victoria, Australia."

By Ernest Lidgey, M.Inst. M.M.

In this paper the author proposes to give a general description of the various deep alluvial leads of Victoria, Australia, in which are the continuations of the rich shallow alluvial deposits which won for Victoria its position as one of the most important gold-producing countries of the world.

When gold was actually first discovered in Victoria is shrouded in mystery, but it was reported to have been found by a shepherd in the Pyrenees in 1849. It was found at Clunes, in March, 1850, near Burnbank, on a tributary of the Loddon River, on the 10th June, 1851; at Mt. Alexander on the 20th July; at Buninyong, on 8th August, and at Ballarat on the 8th September in the same year.

Licenses to dig were first issued on the 1st September, 1851.

During the "fifties" and early "sixties" the alluvial gold mines were the mainstay of Victoria. The attractions of the yellow metal brought a continuous stream of men anxious to enrich themselves.

At first the gold was found on or near the surface, the earth being panned off with rich results; then the miner had to sink shallow holes to obtain gold. Gradually it was found that in following the runs of gold it was necessary to sink deeper shafts, and carry drives or levels along the gutter; water became troublesome, and in many cases it was found that the leads trended beneath flows of basalt, under which it was considered that there was no possibility of working. Some miners more adventurous than the rest, after a time sank a shaft through the basalt or "bluestone," striking payable wash-dirt, thus inducing others to follow into still deeper ground, until now work is being carried on at a depth of over 500 feet beneath basalt and troublesome drift.

The alluvial deposits or placers comprise sands, gravels, or any loosely coherent or non-coherent beds containing gold. (In some cases the sands and gravels have been cemented together with

silica, sulphide of iron, or carbonate of lime.) They may be classified under two groups :—

1. Shallow deposits occurring in the beds of gullies, and in or near existing rivers.

2. Deep leads forming the beds of ancient rivers, which have since become covered by the accumulation of drift or volcanic rocks.

The shallow deposits consist of sand, gravel, loam, and clay, accumulated by the action of existing rivers and streams, often not extending more than 10 or 15 ft. from the surface. They contain gold in fragments of all sizes, ranging from the finest dust to nuggets weighing thousands of ounces.

They occur in river bars and banks, in river flats or inundation plains, in the dry beds of streams which only flow after heavy rains (gullies); in terrace gravels high above the present level of the water (terrace washes); and on the ridges, slopes, and tops of hills. The gravels may contain boulders of any size up to several feet in diameter, or may shade off to fine sand; while sandy clays, especially if on the bed-rock, are frequently very rich. The gold may occur in the grass roots, near the surface of the gravels, or disseminated through the whole thickness of the wash. More commonly, however, the lowest part of the beds just above the bed-rock is richest. The depth at which bed-rock is found varies much; it may crop out at the surface, or in connexion with the deep leads it may be buried beneath hundreds of feet of gravel or basalt.

The portion of the earth known as wash-dirt from which the gold is obtained is usually from a few inches to 3 or 4 ft. in thickness, in rare cases 10 or 12 ft., consisting of clay, sand, and gravel resting immediately on the bed-rock. It sometimes happens, though, that there are two or more beds of wash-dirt, separated by beds of clay. The clay bed on which the upper wash rests is called a "false bottom." When the bed-rock is soft some gold is found in the cracks and crevices, a couple of feet of the bottom often being taken out and puddled with the wash.

It requires no great stretch of imagination to carry one's thoughts back through long ages to the time when what are now the "deep leads" of Victoria formed the main drainage channels of the country. The hills were then much higher than they now are, larger streams of water were flowing in their channels with heavy floods during the rainy season. The animals then alive differed but slightly from those now seen in the bush; the plant life varied more. It is easy to imagine, for instance, the creeks flowing from Creswick northward, forming eventually a river trending nearly

parallel with what is now the Lodden. If there had been any miners in those days, all the gullies and creeks would have been turned over for the gold they contained, wing dams would have been built across the rivers, and the gold obtained from the sand and gravel of their beds. Fortunately for us the miner had not yet appeared upon the scene.

During what is called the Pliocene period, instead of the peaceful scene depicted, a change took place; earthquakes shook the land; volcanoes poured forth their streams of molten lava, filling gullies, creeks, and rivers, in many cases covering the tops of the hills, forming one vast almost level plain of what is now commonly called bluestone or basalt. The same change took place over many other portions of Victoria, especially throughout the south-western district.

The deep leads of Chiltern, Indigo, and some other districts have been covered in a different manner. In this case there has been a slow subsidence of the land beneath the sea, the drift caused by denudation gradually covering up all traces of the old creeks and rivers. Then after a lapse of time the land began slowly to rise, and what was the sea bottom became exposed, forming what is now the surface of the land.

These "deep leads" or old river beds follow in general a very similar course to the present main drainage channels of the colony, although in places they vary much. They occur principally in the central and western portions of the colony, where geological conditions have been favourable for the covering of the wash and its golden contents.

Starting with the westernmost gold-field:—

Stawell or Pleasant Creek.

The deposits of wash containing the gold here have been for the most part formed under what are now hills and elevated surfaces, the modern valleys and flats being usually too poor to pay for working. The principal leads were the Deep Lead and its tributaries, situated from 3 to 5 miles north-west of the town, and the Commercial-street lead and its tributaries, commencing at the reefs, and terminating at Seventy-foot Hill, about 3 miles west of the town. The Great Western lead has been worked for a distance of 2 miles, and a width which exceeds in places 1,200 ft. The gold occurs in fine scales, pieces of 1 dwt. being rare, the largest nugget obtained hardly exceeding 5 dwt. It is estimated that over £100,000 worth of gold was obtained from 2 miles of this lead.

ARARAT.

Here all the shallow and worked-out leads of the field empty into a great main lead system, covered by basalt, from 250 to 300 ft. deep, trending down the valley of the Hopkins, crossing and then following the concealed continuation southerly of the same auriferous belt of rocks, whence the leads worked to the north obtained their gold supplies. Two series of bores have been put down. The northern one proved deep ground at 238 ft. The second series of bores, about 3 miles further south, proved two gutters, one with 10 ft. of wash, and the other with 4 ft. 6 in. of wash. In one of his reports the former Government geologist, Mr. R. A. F. Murray, says: "These borings, coupled with the geological evidences, incontrovertibly prove the existence of a great trunk lead extending within defined limits for at least 8 miles, and probably a considerable distance further, beneath the basalt to the south."

LANDSBOROUGH.

At Landsborough a lead has been traced for a considerable distance. It is tortuous, the sinking somewhat difficult. Many of the gullies in Landsborough, Barkly, and Navarre, have been very rich.

BEAUFORT.

The main lead at Beaufort, formerly known as Fiery Creek, has been worked from its rise at the head of Fiery Creek to the town of Beaufort, a distance of about 6 miles. This lead trends through the township in an easterly direction, when it junctions with the Waterloo lead. From the junction the lead will probably trend easterly through Windermere, and junction with the extension of the Haddon lead.

SMYTHESDALE.

The Linton lead, which rises in the Hard Hills, was discovered in 1854, and trends to the south until it junctions with the Standard lead, which was noted for the great depth of black clay met with. In some claims it was 20 ft. in thickness, and contained throughout large quantities of leaves, grasses, and other vegetable substances. Beyond the junction it is known as the Happy Valley lead, which takes a sharp turn to the east and falls into the main Smythesdale lead. This lead rises near Nintingbool and falls southward, junctions with the Happy Valley lead, and from thence trends southward

through Piggoreet, Cape Clear, and on to the Pitfield plains. There are many tributaries to these leads, most of which have been very rich. The continuation of the deep ground to the south of the Stanley, near Cape Clear, has not yet been proved, although it would appear as if the Victorian Mint Company were near it.

From the Hard Hills the Snake Valley lead falls northward, eventually junctioning with the Preston Hill lead. In the Magnum Bonum claim of 12 acres, at a depth of 102 ft., with 16 men, 6,639 oz. 7 dwt. 17 gr. of gold were obtained. The continuation of these two leads will probably join with the Haddon lead.

The Haddon lead rises at Nintingbool and falls northwards, and will eventually junction with the Midas leads.

BALLARAT.

The city of Ballarat is now reached with its main system of leads, fed by tributaries from Little Bendigo, Canadian, and Hiscocks. The records of the returns obtained from them were very great. For instance, during September, 1851, 567 men, with 143 cradles, secured an average of 1 oz. 6 dwt. 9 gr. of gold per man per day. Two men washed 10 lbs. weight of gold before breakfast from two dishfuls of dirt. Two brothers, named Kavanagh, obtained £1,600 worth of gold in one day from a small hole; and when, a little later on, a gold nugget weighing 2,195 oz. was found by a small party of men, no one will wonder that Australia bordered on madness. There are no things so alluring as the discovery of big nuggets of gold, and Ballarat was famous for its nuggets. Amongst the most valuable were the "Welcome," found at Bakery Hill, which realised £9,325; the "Lady Hotham," at Canadian Gully, £3,000; the "Nil Desperandum," £1,050; and another at Canadian Gully, £5,532.

In 1855 the well-known leads of Inkerman, Red Streak, and Frenchman's, were opened; whilst in 1856, Esmond's, Malakoff, Milkmaids, and Redan were worked. Portions of the leads were very rich. As much as 1,637 oz. of gold were obtained for one day's washing at the Band of Hope Company. It is estimated that from 20 mines, gold to the value of £5,902,050 has been won, and that dividends to the extent of £2,457,496 have been paid, although the total calls only amounted to £594,914.

ROKEWOOD.

Some 20 miles south of Ballarat is situated the well-known Break-o' Day lead, which falls into and through the town of Roke-

wood, in and around which town a large amount of prospecting has lately been carried out in searching for a continuation of the lead.

Upper Bet Bet Valley.

The Upper Bet Bet Valley system extends from Lexton through Caralulup, Lillicur, and Bung Bong, to Rathscar, and the Avoca Valley. I can hardly do better than describe it in the words of Mr. Murray, in a report forwarded to the Agent-General for Victoria:—

"The general course of the main trunk lead is unmistakably marked by the strip of basalt, from 1 to 2 miles in width, which follows the valley, and is bounded on the east and west by ranges of outcropping silurian rock.

"Four distinct series of borings with the diamond drill have furnished most conclusive evidence as to the existence of a large and important trunk lead—the gravel deposit of an ancient river bed—now overlaid by basalt, and corresponding in general direction to that of the upper portion of the existing Bet Bet Valley.

"In the southern portion, at the northern boundary of the parish of Lexton, the borings proved the deepest ground to be 441 ft. from surface to the silurian bed-rock, with 8 ft. of heavy gravel resting on the latter.

"Two miles northwards on the course of the lead, the borings proved a depth of from 260 to 270 ft., with heavy gravel, from which prospects of gold were obtained.

"Five miles further north, in the parish of Lillicur, the deep gravel on the bed-rock was struck at 240 to 248 ft., and 7 miles still further northwards in Bung Bong, a line of bores at short intervals proved the lead from side to side with great exactness, the deepest ground being 330 ft. from surface to bed-rock, and containing over 20 ft. in thickness of gravel.

"The course of the lead goes northward, about 3½ miles into Rathscar, where it turns westward, and in about 2 miles further is joined by the Homebush lead. Beyond here it passes into and down the Avoca Valley, where it is joined by the Avoca lead system.

"For about 20 miles down the Avoca Valley, the alluvial deposits are very wide, and the true position of the lead has not been proved, though its existence is indisputable; but at Archdale, beyond the limits of the area shown on the map, recent borings at a place where the valley contracts, have proved the deep ground with heavy wash from which gold prospects were obtained. Beyond here, again, the valley continues for several miles to beyond St. Arnaud.

"We have thus an aggregate length of over 40 miles of a main trunk lead traversing auriferous rocks, fed by various rich tributary leads, and with numerous rich shallow workings in the bordering country on either side. That this lead has not yet been tried may well be a matter of surprise; but for a long while the attractions of the leads of the Ballarat, Creswick, and other districts, already well known, prevented much attention to exploratory work. Subsequently, the financial depression for a time paralysed industry, but recently a more healthy tone has prevailed, and active steps are being taken to develop the leads.

"There is no doubt that heavy water will have to be contended with, necessitating very powerful and costly pumping machinery; but there exists no doubt in my mind that the lead system above described will prove highly auriferous, and will be remunerative if it be properly worked. The portion extending from the southern line of bores at the boundary of Lexton to the Homebush leads, a distance of 20 miles, without reckoning sinuosities, deserves special attention, being so well defined, so well proved by borings, and being bordered by such richly auriferous country, indicating that the bed-rock beneath the lead contains auriferous matrices which have furnished the alluvial gold supply in the gravels resulting from their degradation."

Loddon Valley.

The Loddon Valley lead, with its tributaries, may be considered as one of, if not the most important, lead system in Victoria.

The Midas and Dowling Forest group of mines form a portion of this system, with a total length of unworked ground between the Dowling Forest mine and the Junction of 16 miles. At the marvellous Berry group of mines, there is a section of 4 miles of unworked ground.

Following to the north from the junction of these two lead systems, as far as the parallel of Carisbrook, the Mount Greenock, and Majorca and Carisbrook leads come in from the west, and the Loddon leads from the east. Here the boring near Moolort has proved a large and well-defined trunk lead, and the Charlotte Plains Mine before being flooded out proved valuable gold-bearing wash; the length of the section is 16 miles, the whole of which follows the auriferous belt. The next section is also a most important one; it is 8 miles long, and extends from Moolort up close to Eddington, where the Bet Bet system joins it. From Eddington to Bridgewater is 20 miles, and in the total length of the system there are 64 miles of unworked leads.

AMHERST, &c.

The leads from Amherst, Daisy Flat, Alma, and Maryborough nearly all trend towards Timor, forming the deep leads worked by the various Duke companies.

SADOWA.

The Sadowa lead, near Talbot, was worked in the shape of a horse-shoe bend to a depth of 80 ft., with a width of 120 ft. At the No. 3 Sadowa the lead was 140 ft. deep, and from here 700 oz. of gold were obtained in a fortnight. It was worked to a wall of rock, and abandoned. Mr. Stirling considers that an intrusive dyke formed the wall of rock which interrupted the lead.

DUNOLLY.

The Dunolly and Burnt Creek leads, with their tributaries, were in the old days very rich. In a report, Mr. Stirling states: "The leads trending down the Bet Bet Valley, such as Chinaman's Flat and Four-mile Creek, as well as that proved by boring near the township of Bet Bet, where good prospects of gold were obtained, all suggest the probability of the extension of these leads, and that where the Bet Bet Creek lead junctions with the main Bet Bet Valley lead rich deposits of gold may be found."

DAYLESFORD.

The Daylesford district has been noted for some of its rich alluvial leads. The stream of lava covering the deep leads can easily be traced. These leads are nearly all above the level of the present stream. Amongst the most noteworthy are the Wombat Hill lead, Dead Man's lead, Italian Hill lead, O'Hara Bourke lead, Fern Hill lead, and Jim Crow Creek.

BENDIGO.

The alluvial gullies at Bendigo have been noted for their richness. They trend towards the Bendigo Creek into a main lead near the White Hills, which has been traced for a length of 7 miles. Below the White Hills several tributaries fall in from the westward. It then trends northwards to the Huntly lead, from which in one claim of 20 acres, £8,000 worth of gold was taken by one party. Subsequently, 13 engines, pumps, and winding plants were erected, and it is stated that £445,000 worth of gold was obtained. It is estimated that for a distance of 6 miles from the White Hills the lead yielded 500,000 oz. of gold, worth £2,000,000.

HEATHCOTE.

The McIvor Creek was worked at Heathcote for some years, most of the gold being found in nuggets, the heaviest of which was 658 oz. Several of the tributary gullies are still being worked by a few men, an occasional nugget being found.

CASTLEMAINE.

Very extensive gullies have been worked in the Castlemaine district at Chewton, Maldon, Castlemaine itself, and down through Yandoit to Newstead.

The Malmsbury and Coliban Valley system of leads heads from near the main dividing range between Trentham and Blackwood.

This lead system is fed by various tributaries from Trentham, Lauriston, Taradale, and Malmsbury. Nothing in the way of mining has been done on the main system from Trentham to Lauriston. Near Carlsruhe a line of bores proved deep ground with 2 ft. of wash at a depth of 348 ft. from the surface. Four and a half miles northward of here, near Lauriston, a well-defined lead was proved at about the same depth, with 7 ft. of wash. Two miles further north a third series of bores proved the greatest depth to be 371 ft., with 2½ ft. of wash. Near Malmsbury the Taradale leads junction with the main lead, and following down the valley of the Coliban a series of bores at Redesdale proved the bed-rock to be 223 ft. from the surface, with 3 ft. to 5 ft. of wash, the fall of the surface from Kyneton to Redesdale being evidently greater than that of the lead.

From Trentham to Axedale, on the general course of the lead system, the distance is fully 50 miles, and from near Carlsruhe to a little beyond Redesdale, where the position of the lead has been proved, is a distance of about 25 miles, sufficient room for a large working population of miners.

PLENTY RIVER.

Near Melbourne a lead rises in the Plenty Ranges. Several gullies and creeks have been worked with satisfactory results, notably Jack's Creek and Deep Creek. In the town of Whittlesea a shaft was sunk, about 20 years ago, through 20 ft. of wash. At South Morang, a tributary of the main lead has been worked for some years by means of tunnels driven from the banks of the river beneath the basalt. Boring is now being undertaken to prove the position of the lead, which probably extends southward towards Collingwood.

TANJIL.

The Tanjil lead is at its upper end, near the township of Tanjil, considerably above the river level. The actual head of the lead, which once probably extended as far as the present river, has been removed, and its course obliterated during the erosion of the latter. It has a more rapid fall than the river, hence the further it is traced the deeper it will be below the surface. The lead trends to the south-east.

NEERIM.

A lead exists at Neerim, which is about 20 miles long, and, according to Mr. Murray, is traceable with ease by surface indications. Another lead also occurs at Moondarra about the same length. These are both very similar to the Tanjil lead.

RUTHERGLEN.

The shallow leads on the Rutherglen field were exceedingly rich, and were practically worked out in a few years. After a lapse of some years, work was begun on the deep leads, of which there are two main ones, the Great Northern and the Great Southern, the latter of which is a continuation of the Chiltern Valley lead. Water is the great difficulty, pumping expenses swallowing up a large part of the working capital. The principal companies now at work are—the Great Southern, Great Southern and Chilton Valley United, Great Southern No. 1, Prentice United, North Prentice, the Great Northern Extended, and the Wahgunyah.

OVENS RIVER.

The Ovens River, with nearly all its tributaries, from the source to below Porepunkah has proved auriferous. The workings have been mostly confined to the easily accessible deposits at shallow depths, either on the bed-rock or on false bottoms, but attention is being now directed to the great masses of gravel too poor for the individual miner, but remunerative if worked on a large scale by means of hydraulic sluicing. From the head of the Ovens to Freeburgh is a distance of 25 miles. There are various shallow workings down the valley to below Myrtleford, but in no case has the deep ground had a fair test, and, as such deep ground must exist, it should be tried at one or two places in order to prove its auriferous character, for which there is every reasonable probability of success.

Dargo High Plains.

The Dargo High Plains leads are situated beneath a basaltic plateau near the head of the Crooked and Dargo Rivers, and are nearly 5,000 ft. above the level of the sea. In the early days the gullies surrounding the plateau were worked up their steep sides to the edge of the tableland. Much prospecting has been done at various times to test whether the gold continued beneath the plateau, with doubtful success, until recently Mr. Ryan undertook the work. He has driven several tunnels beneath the basalt, and has apparently discovered that a "deep lead" does exist, with every probability of its proving payable. The opinions of the various experts who have visited the field vary much as to the future prospects, but should Mr. Ryan succeed in proving the lead payable, he will open up a large area of practically new country.

Moliagul, etc.

Although not deep leads, I cannot omit mentioning the remarkably rich shallow ground of Dunolly, Moliagul, Tarnagulla, Kingower, Kurting, and Inglewood. It was at Moliagul that the largest gold nugget ever known was found. It weighed 2,280 oz. 10 dwt. 14 grs., and was sold for £9,534.

There are also the more or less rich workings of the Wonnangatta, Humffray, Wongungarra, Crooked, Dargo, Wentworth, Nicholson, and Mitta Mitta Rivers, Boggy Creek, and Haunted Stream.

Many rich alluvial finds have been made on the Delegate, Bonang, and Bemm Rivers, in East Gippsland, and, according to Mr. Hunter, one of the officers of the Geological Survey, there is a probability of a lead being found beneath a flow of basalt, between Bonang and the Jingelalla Rivers.

Unworked Leads.

Although a large amount of work has been done on our alluvial leads, there yet remains an immense amount of proved ground unworked.

The following leads have been proved and remain to be worked:—

	Miles.
Great Western Valley	2
Madame Hopkins, Ararat	8
Landsborough	4
Glenpatrick	4
Beaufort, continuation of	18
Pitfield Plains	5
Rokewood	3
Cardigan	30
Upper Bet Bet Valley	42
Lodden Valley	64
Spring Hill Leads	20
Malmsbury and Coliban Valley	50
Plenty River	12
Neerim and Moondarra	40
Ovens River, below Myrtleford	10
Rutherglen, proved 20, worked 5	15
Dargo High Plains	20
Making a total of unworked leads of	347

And if we add the minor leads of Huntly, Dunolly, Morrisons, Elaine, Bullengarook, Blackwood, and Lerderberg, Eldorado, Durham, Benalla, Merton, Broadford, and Jingelalla, we have a grand total of nearly 400 miles of unworked leads.

Accessibility.

With the exception of the Dargo High Plains, the whole of the leads mentioned are intersected by railways or are within reasonable distance of railway communication, by good roads.

Water.

As a rule the quantity of water in the deep leads is considerable, necessitating heavy pumping machinery. When in full working order a large proportion of the water pumped is used for puddling and other purposes. Mines near the centres of activity, such as Ballarat, Bendigo, &c., can obtain water for boilers at a cost of 1*s.* per 1,000 gals. from the reservoirs of the Commissioners of Water Supply.

Machinery.

At first the machinery employed in alluvial mining was very crude, often only consisting of a cradle, shovel, and tin dish. Then

the puddling tub was brought into use, and afterwards the puddling machine worked by a horse. Now many of the miners wash their material in iron puddling machines worked by steam. They also require extensive and often expensive winding and pumping machinery.

The Spring Hill and Central Leads Co. at Smeaton have a 400-H.P. triple expansion condensing pumping engine, high and intermediate cylinders tandem, low pressure cylinder alongside, with heavy pumping gear, bobs, sweep-rods, &c.

The Duke United Co. at Timor, have a 400-H.P. triple expansion condensing pumping engine, with surface condenser, the three cylinders side by side, three Cornish boilers, 150 lb. working pressure, &c.

There is now being erected at the Southern and Prentice United Co., Rutherglen, a 400-H.P. triple expansion condensing pumping engine, three cylinders side by side, heavy pumping gear, bobs, travellers, sweeps, and connecting rods, &c. One pair 18-in. coupled winding engines 42-in. stroke, winding gear, 9-feet drums, pit-head pulleys, &c. One 14 in. by 36 in. capstan engine with all gear. Three Cornish boilers 28 ft. by 6 ft. 6 in., 150 lb. working pressure, &c.

There is also being executed at one of the shafts of the Victorian Gold Estates Co., a 400-H.P. triple expansion pumping engine with condenser, heavy pumping gear, steel bobs and travellers, sweep rods and connecting rods, 600 ft. of 26-in. steel pumps, all strapping plates and bolts, 26-in. draw lift workings and four sets of 26-in. plunger workings. One pair 18 in. by 42 in. coupled winding engines, with winding gear, 9-ft. drums, pit-head pulleys, &c. One 14 in. by 36 in. capstan engine and gear. Three Lancashire boilers, 150 lb. working pressure, with all mountings, feed pumps, steam pipes, &c., complete,

A large proportion of the pumping machinery in use is made in Ballarat, and the remainder in Melbourne, Bendigo, Castlemaine, and Stawell.

Cost of Working.

It is impossible from the balance sheets issued by the different companies to obtain any information as to the detailed costs of shaft sinking, driving, blocking, puddling, &c. The balance sheets are so arranged as to show, for instance, the total cost of labour, which will include all progressive work, total cost of ironmongery, firewood, &c., but in no case are the accounts so divided as to show what amount of either labour or materials have been used on any particular portion of the work.

The following table shows the yield of gold, amount paid in dividends, capital paid up (working capital), and percentage of dividend on working capital on 21 mines, for the year 1897 :—

Company or mine.	Locality.	Yield of gold.		Dividends.	Capital paid up.	P. cent. of dividend on working capital.
		oz.	dwt.	£	£	
Chalk's No. 1	Maryborough	7,025	1	1,500	81,000	1·85
North Prentice	Rutherglen	5,309	12	1,000	51,333	1·95
Hepburn Estate	Creswick	7,134	3	3,600	92,100	3·91
Berry Consols Extd.	,,	8,974	15	5,000	75,500	6·62
Barambogie	Chiltern	7,047	14	2,000	20,000	10·00
Great Southern No. 1	Rutherglen	4,915	1	2,000	19,067	10·49
John Woods	Stawell	2,604	14	625	5,625	11·11
Chalk's Freehold	Maryborough	2,043	18	5,000	44,500	11·24
Kempson's	Stawell	927	13	500	4,125	12·12
Union	Pitfield	2,256	8	695	5,000	13·90
Prentice Freehold	Rutherglen	8,731	0	7,500	29,250	25·64
Berry Consols	Creswick	17,679	17	20,500	78,000	26·28
Chalk's Junction	Maryborough	2,774	13	1,200	4,000	30·00
Pitfield Plateau	Pitfield	2,162	4	600	1,350	44·44
Homeward Bound	Stawell	1,178	8	2,206	2,500	88·24
Lord Brassey	Rokewood	529	5	1,528	1,400	109·14
Madame Berry West	Creswick	15,041	12	10,350	9,000	115·00
Great Northern Extd.	Rutherglen	13,137	19	32,500	20,000	162·50
Hanlon	Rokewood	4,254	7	3,150	1,518	207·51
Carr's Freehold	,,	1,191	14	1,190	420	283·33
Laidler's Freehold	,,	1,625	6	3,600	1,000	360·00

The following lists of mines in the Creswick and Carisbrook districts, not now working, show good records :—

Gold Won and Dividends Paid from some of the Creswick, Kingston and Smeaton Mines, as ascertained from Official Records

Name of company.	Gold won.			Value.			Rate of Royalty.	Royalty.			Dividends.			Capital called up.		
	oz.	dwt.	gr.	£	s.	d.	per cent.	£	s.	d.	£	s.	d.	£	s.	d.
Madame Berry	387,313	17	12	1,588,515	12	2	7½ & 9	128,317	12	6	855,450	0	0	15,975	0	0
Lone Hand	126,050	1	12	517,330	6	1	7½	38,799	15	11	244,450	0	0			
Ristori	104,224	10	12	430,918	16	4	7½	32,153	14	2	199,500	0	0	6,650	0	0
Ristori West	38,491	5	0	157,970	14	7	7½	11,847	15	11	73,000	0	0	12,000	0	0
Earl of Beaconsfield	24,013	0	0	98,553	7	1	7½	7,391	8	9	20,100	0	0	32,700	0	0
Loughlin	38,176	0	0	154,771	17	4	7½	11,607	17	9	58,650	0	0	17,400	0	0
New Australasia	90,203	0	0	374,342	9	0	...	...			98,250	0	0	11,250	0	0
Dyke's Freehold	39,447	17	10	161,896	4	7	10	16,189	12	5	53,231	0	0			
Hepburn Home Paddock	30,315	6	20	124,510	14	6	...	9,286	17	6	36,900	0	0	9,200	0	0
Baron Rothschild	13,778	1	6	56,547	8	9	10	5,654	14	10	33,100	0	0			
Lewers' Freehold	13,934	0	0	57,203	17	6	10	5,720	6	9	32,800	0	0	400	0	0
Cameron's	15,652	5	0	64,239	8	10	7½	4,817	19	3	39,975	0	0			
De Murska	18,576	6	0	76,240	4	7	7½	5,718	0	4	28,200	0	0			
Robinson's	10,104	12	4	41,470	19	11	...	...			21,075	0	0			
Richardson's Western	13,012	18	5	53,407	3	3	10	5,340	14	3	20,000	0	0			
Cunning's	9,260	15	0	38,007	13	2	10	3,800	15	3	19,176	0	0			
Ryan's Junction	11,962	1	18	49,094	8	0	7½	3,681	19	7	16,050	0	0			
Kingston Park	13,854	12	0	56,869	19	4	7½	4,265	4	11	15,800	0	0			
Bunyan's	8,818	18	0	35,194	4	9	7½	2,639	11	4	14,866	0	0			
Hawkins'	5,793	4	5	23,776	5	10	10	2,377	12	7	9,283	0	0			
Australasian Eaglehawk	11,623	11	0	47,704	19	8	7½	3,566	17	5	7,350	0	0			
Lewers' Western	3,262	0	0	13,387	15	10	10	1,338	15	7	6 000	0	0			
Spring Hill Reserve	1,851	15	0	7,599	17	3	...	...			4,000	0	0			
Grey's	2,283	10	0	9,171	17	3	10	917	3	8	3,150	0	0			
Australasian Extended	8,574	18	15	35,192	19	5	7½	2,639	9	5	*			26,775	0	0
Davies Junction	7,873	12	0	32,314	11	3	7½	2,423	10	10	*					
Lord Harry	6,850	6	13	28,114	17	6	7½									

* Dividends could not be ascertained.

Majorca and Carisbrook Returns.

	oz.	dwt.	£	s.	d.	Dividends. £	s.	d.
Sadowa	50,138	0	205,565	0	0	Not ascertainable.		
New Kong Meng	96,727	0	377,321	0	0			
Napier Freehold	72,698	0	278,746	0	0			
Goldbrook	12,861	0	52,730	0	0			
Woolbrook	2,642	0	10,832	0	0			
Carisbrook United	8,261	0	33,870	0	0			
Union	20,486	0	83,992	0	0			
Hoffnung	25,000	0	102,500	0	0			
All Nations	15,000	0	61,500	0	0			
*Black Ball	10,000	0	41,000	0	0			
*Talbot Lease	10,000	0	41,000	0	0			
*Perseverance	9,000	0	36,900	0	0			
*Oriental	8,000	0	32,800	0	0			
*German Tunnel	8,000	0	32,800	0	0			
*Calley's Tunnel	6,000	0	24,600	0	0			
*Darling	4,000	0	16,400	0	0			
*Nicholls' Freehold	6,500	0	26,650	0	0			
Mount Greenock Estate	4,000	0	16,400	0	0			
South Greenock	3,132	0	12,841	0	0			
Royal Standard	2,600	0	10,660	0	0			
Brunswick	1,934	0	7,929	0	0			
Rip Van Winkle	1,836	0	7,527	0	0			
Perseverance	1,813	0	7,433	0	0			
	380,628	0	1,521,996	0	0			

* Co-operative parties.

Supplies.

The following gives the Melbourne prices of many of the items required in mining. Prices in Ballarat, Bendigo, Castlemaine, Maryborough, are usually a little higher, although in large contracts a reduction of these prices can be obtained.

Prices of Mining Supplies in Melbourne.

Anvils, up to 56 lb., 40*s.*; 57 to 84 lb., 34*s.*; over 84 lb., 30*s.* per cwt.

Axes, American, 42*s.* to 60*s.* per dozen.

Belting, leather, Colonial, 40 per cent. off list.

" " English, at list prices.

" India-rubber, 20 per cent. off list.

Blocks, Weston's, patent, with 40 ft. chain, 1 ton, 40*s.*; 2 tons, 62*s.* 6*d.*; 3 tons, 105*s.* each.

Bolts and nuts, round head, square nut and shoulder, $\frac{3}{8}$-in., 32*s.*; $\frac{1}{2}$-in., 26*s.*; $\frac{5}{8}$-in., 24*s.*; $\frac{3}{4}$-in., 22*s.* per cwt.

Candles, 30 lbs. in a box, 5½*d.* per lb.

Copper sheets, 9½*d.* per lb.

Cotton waste, 36*s.* per cwt.

Chain (short link), $\frac{1}{4}$-in., 28*s.*; $\frac{5}{16}$-in., 26*s.*; $\frac{3}{8}$-in., 24*s.*; $\frac{7}{16}$-in., 22*s.*; $\frac{1}{2}$-in., 20*s.*; $\frac{5}{8}$-in., 18*s.*; $\frac{3}{4}$-in., 18*s.*; $\frac{7}{8}$-in., 18*s.* per cwt.

Dishes, gold prospecting, 1*s.* 9*d.*, 2*s.* each.

Explosives, blasting powder, 5½*d.* per lb.
- „ detonators, No. 5, 3*s.* 6*d.*; No. 6, 4*s.* per box.
- „ dynamite (imported), 82*s.* 6*d.* case of 50 lbs.
- „ fuse, Bickford's D.T., 9½*d.* per coil.
- „ gelignite, 77*s.* 6*d.* per case of 50 lbs. (imported).

Files, taper saw, 4½-in., 3*s.* per dozen.
- „ half-round bastard, 12-in., 11*s.*; 14-in., 15*s.* per dozen.
- „ flat „ 12-in., 11*s.*; 14-in., 15*s.* „

Grease, anti-friction, 10*s.* 6*d.* per cwt.

Grindstones, 22-in., 5*s.* 6*d.*; 24-in., 8*s.*; 30 in. by 4 in., 16*s.*; 34 in. by 4½ in., 27*s.* 6*d.*; 36 in. by 5 in., 32*s.* 6*d.* each.

Hammers, steel sledge, 9*d.* per lb.

Handles, axe, 9*s.* 6*d.* and 11*s.* per dozen.
- „ D shovel, 12*s.* per dozen.
- „ hammer (36-in.), 8*s.* 6*d.* per dozen.
- „ pick, 9*s.* per dozen.

Hose (india-rubber) cut lengths, 1-in., 1*s.*; 1½-in., 1*s.* 6*d.*; 2-in., 2*s.* per foot.

India-rubber (sheet), 2*s.* per lb.

Iron bar, £9 10*s.* per ton
„ black sheet, £11 per ton
„ Low Moor, £26 per ton
„ W.R.O., £12 10*s.*
} Plus usual trade extras for very small and very large sizes.

Iron, galvanized, corrugated, 5-, 6-, 7-, 8-ft. lengths, £16 per ton.
„ „ „ 9-ft. „ £17 „
„ „ „ 10-ft. „ £18 „

Lead, red, in 7 lb., 30*s.*; in 14 lb., 26*s.* per cwt.

Lead, white, in 28 lb., 28*s.* 6*d.*; in 112 lb., 28*s.* per cwt.

Nails, Ewbanks, 2-in., 37*s.*; 2½-in., 32*s.* 6*d.*; 3-in., 31*s.* 9*d.*; 4-in., 26*s.* 6*d.*; 6-in., 25*s.* per cwt.

Nails, wire, 1 in. by 16, 28*s.* 6*d.*; 1½ in. by 13, 23*s.*; 2 in. by 10, 19*s.*; 2½ in. by 9, 18*s.* 6*d.*; 3 in. by 8, 18*s.*; 6 in. by 4, 17*s.* 6*d.* per cwt.

Broken lots, 2½*d.* to 3*d.* per lb.

Oil, colza, in 5-gal. drum, 3*s.* 9*d.* per gal.

Oil, castor, in 17-gal. cases, 3*s.* 6*d.* per gal.
„ kerosene, in 8-gal. cases, 9¼*d.* „
„ linseed, boiled, in 5-gal. drums, 2*s.* 9*d.* per gal.
„ „ raw „ „ 2*s.* 7*d.* „
„ machine black „ „ 1*s.* 6*d.* „
„ „ engine „ „ 2*s.* 9*d.* „
„ neat's-foot „ „ 2*s.* 3*d.* „
„ olive „ „ 2*s.* 7*d.* „
Packing flax, 10*d.* per lb.
Paint, mixed, in 2-lb. tins, 8*s.* per dozen tins.
Picks, miners, single end (Hillmans), 4, 2*s.* 9*d.*; 4½, 3*s.*; 5, 3*s.* 3*d.* each.
Picks, double end, 3*s.* 3*d.* each.
Pipe, gas, ½-in., ¾-in., 1-in., 1¼-in., 1½-in., 2-in., 2½-in., 3-in., 4-in., 4¼*d.*, 6*d.*, 8½*d.*, 1*s.*, 1*s.* 3*d.*, 1*s.* 9*d.*, 3*s.* 3*d.*, 4*s.* 6*d.*, 7*s.*, per foot. Black, less 72½ per cent. Galvanized, less 62½ per cent.
Pulleys, wood, single 5*d.*, double, 10*d.* per inch.
Quicksilver, 2*s.* 2*d.* per lb.
Retorts, 4-in., £2 10*s.*; 6-in., £3 10*s.*; 8-in., £4 10*s.* each.
Rivets, pan head, ¼-in., 35*s.*; $\frac{5}{16}$-in., 34*s.*; ⅜-in., 33*s.*; ½-in., 25*s.*; ⅝-in., 18*s.*; ¾-in., 17*s.*; ⅞-in., 17*s.* per cwt.
Rivets, copper, 1*s.* per lb.
Rope, steel, varies according to quality and maker.
„ Manilla, above 1 in. circum., 49*s.* per cwt.
„ tarred „ „ 50*s.* „
Saws, cross-cut, lightning, 2*s.* 3*d.* per foot.
„ „ peg-tooth, 1*s.* 9*d.* and 2*s.* 3*d.* per foot.
„ hand, from 2*s.* to 6*s.* each.
Shovels, D-handled, round point, "Ames'," 4*s.*; "Day's," 3*s.*
„ long „ „ „ „ 4*s.*; „ 3*s.*
Steel, oct. drill, 29*s.* 6*d.* per cwt., cast tool, 42*s.* per cwt.
„ Stubbs' cast tool, 9*d.* per lb., double shear, 42*s.* per cwt.
„ blister, 24*s.* per cwt.; spring, 13*s.* per cwt.
Washers, iron, 20*s.* per cwt.

All prices subject to market variations.

Cost of Charcoal, &c., in various Mining Centres.

	Ararat.	Ballarat.	Bendigo.	Chiltern and Rutherglen.	Maryborough.
Charcoal	20*s.* per ton	7*d.* per bag	20*s.* per ton	18*s.* per ton.	1*s.* bag.
Coal (per ton)	*	19*s.* 3*d.* to 17*s.* 9*d.*	21*s.* to 32*s.* 6*d.*	*	30*s.*
Coke (per ton)	...	26*s.*	25*s.*	...	27*s.* 6*d.*
Firewood (ton of 50 ft.)	5*s.*	5*s.* 9*d.* to 6*s.* 8*d.*	3*s.* 6*d.* to 7*s.* 6*d.*	4*s.* 9*d.* to 6*s.*	5*s.* to 6*s.* 6*d.*
Laths (per 100)	13*s.* 6*d.*	10*s.* 6*d.* to 11*s.*	11*s.*	13*s.* to 16*s.*	12*s.*
Props (per 100 ft.)	8*s.* to 16*s.*	3*s.* 6*d.* to 16*s.*	8*s.* to 13*s.*	10*s.* to 20*s.*	3*s.* 3*d.* to 16*s.*
Sawn timber (hard wood, 100 ft. super.)	11*s.*	5*s.*	12*s.* 6*d.*	10*s.* to 12*s.*	5*s.* 6*d.*

* Not used.

The following is a list of salaries and wages paid per week for different kinds of labour in each gold-mining district, as given in the Annual Report of the Secretary for Mines, Victoria, 1897 :—

District.	General manager.	Legal manager.	Mining manager.	Engineer.
Ararat	£5	£1 to £2 10*s.*	£3 10*s.* to £5 10*s.*	£2 10*s.* to £4 10*s.*
Ballarat	£6	£2 to £5	£3 to £6	£3 to £4
Beechworth	£3 to £9	10*s.* to £3	£3 to £6	£2 10*s.* to £5
Castlemaine	£3 to £6	15*s.* to £3	£3 to £5 10*s.*	£2 15*s.* to £3
Gippsland	£2 10*s.* to £6	£1 to £3	£3 to £6	£2 10*s.* to £4 10*s.*
Maryborough	£3 to £7	£1 to £3	£2 10*s.* to £3 10*s.*	£2 10*s.* to £3
Sandhurst	£3 to £9	5*s.* to £3	£2 10*s.* to £4 10*s.*	£3 to £4

District.	Engine driver.	Pitman.	Blacksmith.	Carpenter.
Ararat	£2 10*s.*	£2 8*s.* to £2 10*s.*	£2 10*s.* to £3	£2 2*s.* to £3
Ballarat	£2 5*s.* to £2 10*s.*	£2 5*s.* to £3 10*s.*	£2 5*s.* to £2 14*s.*	£2 5*s.* to £3
Beechworth	£2 5*s.* to £2 16*s.*	£2 5*s.* to £2 10*s.*	£2 5*s.* to £3.	£2 5*s.* to £3
Castlemaine	£2 5*s.* to £2 10*s.*	£2 5*s.*	£2 5*s.* to £3.	£2 5*s.* to £2 14*s.*
Gippsland	£2 5*s.* to £3	£2 to £2 15*s.*	£2 5*s.* to £3.	£2 5*s.* to £2 10*s.*
Maryborough	£2 10*s.*	£2 10*s.* to £3	£2 10*s.* to £2 15*s.*	£2 10*s.*
Sandhurst	£2 10*s.*	£2 5*s.* to £3	£2 10*s.* to £3	£2 5*s.* to £3

District.	Foreman of shift.	Miners.	Surface men.	Boys.
Ararat	£2 10*s.* to £2 15*s.*	£2 to £2 15*s.*	£1 10*s.* to £2 5*s.*	£1 to £1 10*s.*
Ballarat	£2 5*s.* to £2 15*s.*	£2 to £2 10*s.*	£1 15*s.* to £2 5*s.*	15*s.* to £1 10*s.*
Beechworth	£2 5*s.* to £2 15*s.*	£2 to £2 10*s.*	£1 10*s.* to £2 5*s.*	£1 to £1 10*s.*
Castlemaine	£2 5*s.* to £2 10*s.*	£2 2*s.* to £2 8*s.*	£1 15*s.* to £2 2*s.*	15*s.* to £1 15*s.*
Gippsland	£2 10*s.* to £3	£2 5*s.* to £2 10*s.*	£1 10*s.* to £2 10*s.*	£1 to £1 10*s.*
Maryborough	£2 10*s.*	£2 5*s.* to £2 10*s.*	£1 10*s.* to £2 2*s.*	15*s.* to £1 10*s.*
Sandhurst	£2 8*s.* to £2 15*s.*	£2 5*s.*	£1 10*s.* to £2 2*s.*	15*s.* to £1 10*s.*

Mining Laws and Labour Covenants.

Leases for mining deep alluvial leads on Crown lands and private property may be granted by the Government of Victoria for any

term not exceeding 15 years, renewable so long as labour covenants are carried out, and the area so granted is, under certain conditions, practically unlimited. Rent payable for Crown lands and private property alienated since 29th December, 1884, 2*s.* 6*d.* per acre per annum, and for private property alienated prior to that date, for any area not exceeding 20 acres, 10*s.* per annum; and for any area exceeding 20 acres, 6*d.* per acre per annum, paid half-yearly in advance.

The labour conditions are as follows :—

	4 acres or under					.....	2	men.
Over	4 acres	and	up to	6	acres	.....	3	„
„	6	„	„	8	„	.....	4	„
„	8	„	„	10	„	.....	5	„
„	10	„	„	20	„	.....	10	„
„	20	„	„	24	„	.....	11	„
„	24	„	„	30	„	.....	12	„
„	30	„	„	40	„	.....	15	„
„	40	„	„	50	„	.....	18	„
„	50	„	„	100	„	.....	30	„
„	100	„	„	150	„	.....	38	„
„	150	„	„	200	„	.....	46	„
„	200	„	„	250	„	.....	54	„
„	250	„	„	300	„	.....	60	„
„	300	„	„	400	„	.....	70	„
„	400	„	„	500	„	.....	80	„

The deep alluvial leads have been and are being worked with satisfactory results; the cost of material and the mining laws and labour conditions are reasonable; the machinery required for pumping is occasionally expensive; but when we remember that the Madame Berry Mine alone obtained from less than 1 square mile of ground gold to the value of £1,588,515 12*s.* 2*d.*, paid £855,450 in dividends, £433,000 in wages, and £130,000 in royalty on a capital of £15,975; and that from the Creswick Mines alone we have obtained 1,343,614 ounces of gold worth £5,513,281 8*s.*, one can imagine the possibilities of the future when the whole 400 miles of now unworked leads shall be in full operation.

REPORTS RELATING TO DEEP ALLUVIAL LEADS OCCURRING IN THE OFFICIAL PUBLICATIONS OF THE DEPARTMENT OF MINES, VICTORIA.

Goldfields and Mineral Districts of Victoria. R. Brough Smyth, 1869. (All leads discovered prior to 1869.)

Quarterly Reports of Mining Registrars and Surveyors, Returns in all issues from 1866 to 1891.

Reports in Quarter ended:—

June, 1870. Paper on the Durham Lead from Sebastopol to Mount Mercer (R. A. F. Murray).

June, 1874. Report on the Freestone Creek Deep Lead, Briagolong (R. A. F. Murray).

Report on the Durham Lead, Buninyong (Robert Etheridge and R. A. F. Murray).

Sept., 1886. Alluvial Gold Nuggets, Report on Sources of (R. A. F. Murray).

Notes on Kingston and Smeaton Mines (R. Allan).

June, 1887. Sources of Alluvial Gold Nuggets (R. M. Sergeant).

History of Napier Freehold Co., Majorca.

" Band of Hope Co., Ballarat.

Sept., 1887. Geological Progress Report of Beechworth (E. J. Dunn).

History of Working Miners' United Company, Homebush (F. T. Outtrim).

Dec., 1887. Report on Dowling Forest and Midas Group of Mines (R. A. F. Murray).

March, 1888. Geological Progress Report on Meredith Nos. 1 and 2 (E. J. Dunn).

Sept., 1888. Report on Avoca and Homebush Mines (R. Allan).

June, 1889. " Creswick, Kingston, and Smeaton Mines (R. Allan).

History of Duke Company's Mine, Timor (C. Smith).

" Daylesford Goldfield (Mark Amos).

June, 1890. Report on the Clunes Goldfield (R. A. F. Murray).

March, 1891. Notes on the Underground Workings of Madame Berry and Berry Consols Companies.

Reports of Progress of the Geological Survey of Victoria.

No. 1. Geology and Mineral Resources of Ballarat (R. A. F. Murray).

No. 2. Progress Report of the Geological Survey of the Stawell Goldfield (N. Taylor).

Progress Report of the Geological Survey of the Ararat Goldfield (F. Krause).

Report on the Freestone Creek Deep Lead (R. A. F. Murray).

Report on the Country intersected by the Durham Lead, Buninyong (R. Etheridge and R. A. F. Murray).

No. 3. Report on the Geological Survey of the Ararat Goldfield (F. M. Krause).

Report on the Geology and Mineral Resources of South-Western Gippsland (R. A. F. Murray).

Report of the Geological Survey of the Stawell Goldfield (N. Taylor).

No. 4. Notes on the Geological Survey of Creswick (F. M. Krause).

Report on the „ „ Learmouth (N. Taylor).

Notes on the Geological Structure of North Gippsland (A. W. Howitt).

No. 5. Notes on the Auriferous Leads of Middle Pliocene Age in the Creswick Goldfield (F. M. Krause).

Geological Survey of Learmouth (Supplementary Notes by N. Taylor).

Report on the Geological Survey of the Clunes Goldfield (N. Taylor).

Notes on the Geological Survey of Daylesford (F. M. Krause).

Report on the Geological Survey of Portions of Dargo and Bogong (R. A. F. Murray).

No. 6. Remarks on Sites near Smeaton and Clunes suitable for Bores for Testing the Relative Depths of the Basaltic Rocks, and for ascertaining the Direction of Deep Auriferous Leads (R. A. F. Murray).

Report on the Proposed Scheme for the Underground Drainage of the Sebastopol Plateau and Durham Lead (R. A. F. Murray).

No. 7. Report on the Means of Testing the Myer's Flat Lead (R. A. F. Murray).

Report as to the Probability of Auriferous Leads being found near Wangaratta, in the Ovens River Valley (R. A. F. Murray).

Report on the probably Auriferous Character of the Country in the Vicinity of Rutherglen and Wahgunyah (R. A. F. Murray).

Reports on a portion of the Plateau situate between the Thomson and the Tyers Rivers, Gippsland (R. A. F. Murray).

Geological Surveyor's Report on the Deep Leads of the Loddon Valley (R. A. F. Murray).

Report on the Geological Survey of the Area containing the

Clunes, Mount Greenock, and part of the Talbot and Amherst Goldfields (R. A. F. Murray).
Bacchus Marsh District (R. A. F. Murray).
Geological Surveyor's Report on the probable Extension of the Stawell Leads (R. A. F. Murray).
No. 8. Report on the Geological Survey of Daylesford Goldfield (N. Taylor).
Report on the Malmsbury and Lauriston Goldfield (E. Lidgey).
Geological Report on the North-Eastern District (N. Taylor).
No. 9. Report on the Leads in the Ballarat District (R. A. F. Murray).
Report on the Extent of the Deep Leads of Victoria (R. A. F. Murray).
Report on the Deep Lead Systems of the Upper Portion of the Bet Bet Valley and of the Malmsbury and Coliban Valley Districts (R. A. F. Murray).
Hopkins Valley Deep Leads (R. A. F. Murray).
Report on the Huntly Alluvial Lead (J. Stirling).
" " Great Western Valley Lead (J. Stirling).
Notes on the Sadowa (Scandinavian) Lead, Talbot (J. Stirling).
The Auriferous Alluvial Deposits of the Ovens and Buckland Valleys (R. A. F. Murray).
The Deep Leads of the Loddon Valley (R. A. F. Murray).

SPECIAL REPORTS ISSUED BY THE DEPARMENT OF MINES.

Report on a Portion of the Avoca Valley Lead (R. A. F. Murray).
Synopsis of Results of Boring on the Langi Logan, Coghill's Creek, and Loddon Valley Lead System (R. A. F. Murray).
Report on the Parish of Avoca (H. S. Whitelaw).
" " Deep Lead Systems of the Upper Portion of the Bet Bet Valley, and of the Malmsbury and Coliban Valley District (R. A. F. Murray).
Report on the Parish of Beaufort (E. Lidgey).
" " Commeralghip (E. Lidgey).
" " Corindhap (E. Lidgey).
Report *re* Deep Alluvial Leads of the Colony (R. A. F. Murray).
Report on the Goldfields of Dargo High Plains, &c. (R. A. F. Murray).
Report on the Daylesford-Bullarook Quarter Sheet (S. Hunter).
" " Parish of Glenmona (J. Stirling).

Report on the Auriferous Country North of Spring Hill, Creswick (R. A. F. Murray).
Report on the Inglewood District (R. A. Moon).
" " Parish of Kingower (R. A. Moon).
" " Malmsbury District (E. Lidgey).
" " Parish of Mindai (E. Lidgey).
" " " Rathscar (H. S. Whitelaw).
" " Goldfields of Yackandandah Valley, &c. (R. A. F. Murray).
Report on Mining at Tanjil (H. Herman).
Report on the Parish of Yehrip (H. S. Whitelaw).
" " " Bet Bet (S. Hunter).
Report of Lecture on the Deep Leads of Victoria (E. Lidgey).

Reports on boring done by diamond drills and otherwise, appear in Reports on Diamond Drills and Water Augers, Nos. 1, 2, 3, and 4; Annual Reports of the Secretary for Mines, 1889 to 1896; Record of Boring Operations for the year 1897.

DISCUSSION.

The President thought that a point of importance about the author's paper was the fact that there was still a large field for future operations in connection with the deep leads of Victoria. The large tract of country represented an immense amount of work for members of their profession, and although the salaries mentioned were small, he hoped that the colony would be able to learn from the mother country, and find that it was profitable to pay higher wages. They knew, of course, that there were several mines in Australia working under very cheap management, looked after by men who had grown up there, and were well acquainted with the local conditions and able to give an excellent account of the properties under their control. He thought, however, that the tendency of the age was to employ a higher class of labour for such purposes, and it was generally admitted in go-ahead countries that it was bad economy to pay too little for a manager's services.

Mr. Lidgey would probably be able to supplement the paper by some remarks with regard to the information on which the deep shafts were sunk, as while the drill holes which were put down established the existence of gravel, they did not necessarily prove anything regarding the value, and certain chances had to be taken,

the results of which depended largely on the local knowledge of the directions of the pay streaks of the gold.

Mr. Cox said that Mr. Lidgey spoke of the leads as being proved for 400 miles; he took it that the author simply referred to the continuity of the channels, and that boring had proved that the gravel existed over that area. He hoped Mr. Lidgey would be able to give them some details as to what gold was found in the borings, and whether there was a continuance of the lead. It was well known that the leads of Australia (he referred more particularly to New South Wales) varied a great deal, and that leads were found which carried gold for some distance, then came barren ground, and afterwards rich deposits further on. He thought that in the case in question the gold had been fed more or less into shoots from leaders crossed by the streams; if such was the case the 400 miles of gravel would not mean perhaps more than 10 per cent. of payable mines in the distance named.

Mr. Lockhart remarked that it would be interesting to know the character of the gold.

Mr. Lidgey, in reply, said that the lowness of the salaries paid to the managers was due to the fact that most of the Victorian mines were directed by men who had themselves been miners; in fact, the management was practically in the hands of the Board of Directors, the manager in most cases being little more than an underground and overground boss, doing what the Directors told him. The Directors knew the country and could do fairly well by paying a low salaried man, although it was becoming more the custom to pay a higher remuneration for a man to manage the mine under the Board of Directors.

With regard to the occurrence of the pay streaks and the continuity of the leads, the area of 400 miles included all probably good country and possibly some bad. If a map of Victoria was marked with the places where gold was found, both in large and small quantities, it would be noticed that with very few exceptions the gold-bearing country ran in regular bands more or less north and south. The width of the bands varied (as was proved by the gold already found) from ½ mile to perhaps 3 or 4 miles. Where the drainage rivers had formed channels through the country and crossed the bands of rich gold bearing country, the gold gradually became poorer as they passed away, and where they entered another band, or perhaps crossed the same band again, the gold contents were greater. As to the continuity of the leads, gold had been found in the bores in the Ascot, the Victorian Gold Estates, the Charlotte Plains, as well as in many others.

Mr. G. Snelus inquired as to the diameter of the bore holes, and whether a solid core was obtained.

Mr. Lidgey stated that he believed the diameter was 2½. Solid cores were obtained except when going through very vesicular basalt. The cores obtained were, in the case of work done by the Government, all kept for reference at the Government offices.

With regard to the nature of the gold, it varied from float gold, generally more or less worn (although he had seen gold which could hardly be distinguished from that obtained by breaking a piece of quartz containing coarse gold), up to heavy nuggets, which were usually very rounded. There had been considerable discussions locally as to the origin of the gold; most people considered that it had been formed from the denudation of the quartz reefs, but many eminent mining engineers insisted that the gold had been deposited *in situ* in the leads themselves. There was a certain amount of reason on their side, although he did not agree with their views. The leads contained a good deal of pyrites.

The President said that he could not help again referring to the salaries paid to the managers. Their profession would be in a very bad way if the Boards of mining companies were always formed of such eminent mining men that they only required the services of an underground manager. He could not help thinking that the practical knowledge which even such men as had grown up with the country had acquired, hardly constituted them good authorities for laying out the best plan of working a very big property.

The President announced that the next paper was that by Mr. Olds, on "Blue Asbestos." The mineral seemed to be remarkable chiefly on account of its colour, and the samples exhibited were intended to show the applicability of the fibre to various manufactures in use.

"Notes on Blue Asbestos."

By H. F. Olds, Assoc. Inst.M.M.

Blue asbestos occurs in Griqualand West, and is mined over an area of 30,000 acres. It differs from the other varieties of asbestos, such as the Italian, Canadian, or Russian, not only in that it is blue in colour, but in being considerably lighter.

Occurrence.—The asbestos is found in veins, seldom less than 2, or more often 4 to 5 in. wide, formed of closely compacted parallel fibres, which run from wall to wall of the vein without break or fault. The grain is very fine, and even in the rough state the fibres are singularly distinct. Several veins are found, always in regular extent, and the fibre always lies at right angles to the sides of the deposit. The enclosing rock is a dark brown shale. The fibres are somewhat elastic, and easily separable by the fingers. The character of the rock varies considerably, some places are soft and some hard; the better quality asbestos occurs in the hard rock.

Colour.—Its colour is a peculiar lavender-blue, and is caused by the large proportion of protoxide of iron it contains.

Composition.—Its composition is given as—

Silica	51·1	per cent.
Protoxide of iron	35·8	„
Soda	6·9	„
Magnesia	2·3	„
Water	3·9	„
	100·0	

Output.—The present output is over 100 tons per month, and there is apparently unlimited capacity for development and increased turn-out as the demand for it increases.

Labour.—Native labour is employed under European supervision. Very little skilled labour is required, the mining being mostly surface work, or by shallow adits driven in the sides of the hills.

Cost.—The smallness of the cost at which the article is obtained forms a striking feature, being only £5 per ton of 2,000 lb.

Qualities.—It has most of the qualities of white asbestos. It is uninflammable, rot-proof, and unaffected by atmospheric influences, and is a non-conducting material. It is stronger than the ordinary asbestos. In "cobbing" it breaks away from the matrix rock with a clean fracture, and without any fragments of the latter adhering.

Uses.—It is a most efficient covering for preventing loss of heat and condensation, and consequently in economising fuel.

The following gives the average results of two sets of experiments made in 1896:—

	I. Bare pipe.	II. Pipe covered with blue asbestos mattress with 1-in. diameter asbestos cord over it.	III. Pipe covered with blue asbestos 1½-in. cord, with small asbestos string between.
Total lb. water condensed per hour	12·225	3·152	3·484
Lb. water condensed per sq. ft. per hour	1·698	0·437	0·484

In the above experiments the average steam pressure equals 95 lb., and the average engine-room temperature equals 57° Fahr. Surface of each pipe equals 7·2 sq. ft. The asbestos can also be used for packings and jointing materials. The wool is capable of being spun into very fine yarn, of great tensile strength, which can be woven into netting, twine, ropes, and cordage of all kinds, as may be seen from the specimens on the table. A composition is also made from the blue asbestos for rendering cement and other materials unattackable by acid liquors or vapours.

In conclusion, the writer would like to point out that his remarks refer exclusively to the *blue* variety of asbestos, although many of the qualities and uses of the white asbestos are the same as that of the blue.

"Notes on Dry Crushing and Cyaniding of Rand Ore."

By FRANKLIN WHITE, M.Inst.M.M.

INTRODUCTION.

THE mechanical preparation of gold-containing materials by dry crushing is no new thing, and the author does not claim originality in its application to Rand ores. The paper is simply an account of the arrangement of plant and a record of the results obtained in dealing with some 22,000 tons of ore at the Luipaards Vlei Estate Company's mine. The author is indebted to the Directors of the Company for permission to make use of the information obtained.

The quantity crushed is sufficient to give a reliable indication of what can be done by dry crushing with rolls; but the work done was mostly of an experimental character, and the tests made were often carried to an extreme point in order that very definite data might be obtained. The cost of the work, and perfectness of extraction were naturally affected by these experiments.

During the first four months, the ore treated was chiefly oxidised, the first month's residues ran 1·6 dwt.; they then decreased to 1·1 dwt. The last four months' work was on pyritic ore, and the residue assay rose to 1·77 dwt., which afterwards dropped to 1·15 dwt.

The crushing capacity of the plant was so much in excess of what was expected, that it only ran at intervals. Such work could not be done economically.

The author does not wish to introduce dry crushing as a patent medicine which will convert hopelessly unpayable ore into a source of incalculable wealth, or to maintain that it will suit all classes of ore, either clayey or friable, brittle or tough, whether the gold is coarse or fine, free or intimately combined with oxides, sulphides, tellurides, &c.; but he is of opinion that it can in the majority of cases do better work at a cheaper working cost and less outlay in plant, than a wet stamps mill.

The introduction of the cyanide process has materially altered the problem which the mining engineer has to solve, and crushing

machinery should now be chiefly designed to produce a product suited for cyanide treatment. There does not appear to be sufficient reason in many cases, why amalgamation should be resorted to at all.

The Rand gold-bearing conglomerate ("banket") is well known, but in order to indicate clearly what class of ore is referred to in this paper, it will be advisable to mention its chief characteristics. It is a sedimentary deposit, consisting chiefly of more or less rounded fragments or pebbles of quartz, practically non-auriferous, often considerably fractured by earth movements and pressure. The interstices between the pebbles are filled with finer sediment or "cement." Iron pyrites is also present in quantities varying from say, 2 to 10 per cent. The gold is generally in the cement, on the outside of the pebbles or in the iron pyrites. Silica has been subsequently deposited in the conglomerate beds, the originally porous layer being converted into a compact mass. Stringers and pockets of quartz are sometimes found near faults and dykes, containing sulphides of iron, lead, copper, and zinc, but are generally very poor in gold. In a few cases, however, pockets of very coarse gold have been found.

The immense quantity of comparatively low grade pyritic rock which exists on the Rand makes it well worth while to endeavour to improve the methods of working, even if it be only by a shilling or two per ton. The gold is not in the quartz pebbles, which form 75 per cent. at least of the ore, so it is not necessary to crush them so fine as would be required when dealing with a reef, where the gold and matrix were formed at the same time and in close combination. It seems possible that a sufficiently high percentage of gold can be won by less expenditure of power and of money by dry crushing and cyaniding than by wet milling, followed by the "three processes," viz., plate amalgamation, cyaniding of sands, and slime treatment.

The abundance or proximity of water is a question of great importance, especially on the Rand.

The author does not desire in any way to minimise the perfection to which wet milling and the accompanying metallurgical processes have been brought by those who have devoted themselves to it. Indeed, it is a difficult task to snatch an increased percentage of extraction or a lower working cost than what is reached in a well appointed modern plant, but there is a growing recognition that something can be done, and with tangible advantage to those who are pecuniarily interested in gold mining. The highest percentage of extraction does not of necessity always indicate the highest profit.

The last few grains may cost as much as they are worth, in which case the labour expended in getting them is lost.

The objections most frequently urged against dry crushing may be summarised as follows:—

1. Wear of surfaces of crushing machinery.
2. Wear of general machinery, shaftings, &c., and injury to health of workmen from the floating dust.
3. Difficulty of crushing and screening ore if damp.
4. Difficulty of crushing the ore fine enough to liberate the gold.
5. Impossibility of cyaniding coarse gold.

The author hopes that this paper will show that many of these difficulties can be overcome, and that the rest do not exist.

Tables of quantities and costs in a somewhat condensed form accompany this paper, and will be referred to occasionally.

Crushing Plant.

The ore from the grizzlies and sorting floors was fed into a No. 4 Gates breaker, and ran from its shoot to a No. 2, which could deal, under these circumstances, with 20 tons per hour. The crushed ore then passed through a revolving screen, and was separated into three sizes, viz., say, $-1\frac{1}{2}$ in. $+\frac{1}{2}$ in., $-\frac{1}{2}$ in. $+\frac{1}{4}$ in., and $-\frac{1}{4}$ in. Each size was collected in a separate bin, the proportions of these sizes, in the case of pyritic ore, being about

	Weight.		Moisture.
Pieces $-1\frac{1}{2}$ in. $+\frac{1}{2}$ in.	53·4 per cent.		0·83 per cent.
" $-\frac{1}{2}$ " $+\frac{1}{4}$ "	23·8 "		2·50 "
" $-\frac{1}{4}$ "	22·8 "		5·00 "
	100·0 "	Av.	2·225 "

The ore was then raised in trucks to its proper mill-bin, the top of which is 25 ft. above the crusher platform, itself 10 ft. above the ground.

The crushers, three in number, viz., a 7-ft. Niagara pulveriser, and Gates rolls, Nos. 1 and 2, 36 in. in diameter, 15-in. face, rest on strong wood framing in the order mentioned, receiving the $+\frac{1}{2}$, $+\frac{1}{4}$, and $-\frac{1}{4}$ in. ore respectively.

The counter shaft for driving the crushers is fixed over head on the ore bin floor. A traveller is over each crusher to facilitate moving the heavy pieces.

The Niagara pulveriser was not fitted with the ordinary cone and exhaust fan used for fine crushing, as it was only required to crush

to $\frac{1}{2}$ inch. The ore after passing under the roller was delivered into a large revolving trommel, which returned the coarse pieces, allowing the $-\frac{1}{2}$ in. to fall through a shoot on to a horizontal belt conveyer, 18 in. wide, running about 350 ft. per minute. The crushed ore from the two pairs of Gates rolls fell on to the same conveyer, and all was carried to the elevator. This machine, 65 ft. between centres, ran at 300 ft. per minute, and consisted of an 11-in. belt, with steel buckets every foot apart. Each bucket would hold 9·12 lbs. of ore when full.

The ore was then discharged into shoots leading to two double octagonal revolving screens, fitted with bumpers, and 10 ft. long, and tapered, the inner screen being 2 ft. × 4 ft., the outer 3 ft. × 5 ft. and 8 ft. long. The centres of the shafts were $19\frac{1}{2}$ ft. above the mill bin floor.

For pyritic ore the screens were covered with woven wire screening, $\frac{1}{4}$-in. mesh being inside, and $\frac{1}{10}$ in. outside. Three products were thus formed, the first, the $+\frac{1}{4}$ in. passing by a shoot, to No. 1 roll bin. On its way, any pieces of iron or steel were picked out by an electro-magnet. The $+\frac{1}{10}$ in. product went to No. 2 roll bin, and the $-\frac{1}{10}$th fell into a trough with a short worm conveyer which carried it back to shoots leading to two similar screens below. These were covered with 200 mesh (say $\frac{1}{14}$ in.) sieving, and gave two products, that which passed through being finished. The coarser portion went to No. 2 roll bin, while the fine portion was taken by a worm conveyer to the "finished ore" storage bin, from whence, when required, it was taken by a 16-in. belt conveyer to the first row of tanks.

The 18-in. conveyer, elevator, and screens were all enclosed. The escape of dust was so little that the exhaust fans and settling chambers were not used after the first two or three days.

As the screens were shut in it was found necessary to take frequent samples of the finished product in order to detect escape of too large particles through holes, or leakage from shoots. Some wire sieving had been varnished. This was objectionable, as the fine dust settled and the holes became choked.

Crushing Capacity.

When crushing free milling (oxidised) ore to $\frac{1}{8}$ mesh, 20 tons per hour could be finished. Pyritic ore could be delivered through 300 mesh at the rate of $8\frac{1}{4}$ tons per hour. None of the crushing machines have been worked up to their full capacity, as the

feeders did not supply the ore fast enough. As the cyanide plant was easily supplied there was no need to alter the feeds.

With reasonable care underground the ore could be mined dry enough to be crushed and screened without artificial drying.

Table A gives the amounts or proportions of ore crushed to different sizes by each machine.

The ore was fed into the pulveriser at an average of 8¾ tons per hour. As the rolls were continually receiving "returns" from the screens, it was not easy to gauge their capacity.

Wear of Rolls and Pulveriser.

The totals are given in Table B. It is slightly in excess of what would be considered good in stamps, but the table shows clearly that had the pulveriser worked as well as did the rolls the wear would have been considerably less. The work of reducing ore to ½ in. maximum is not hard, and the pulveriser should have done better. The defect did not consist in excessive wear, but in defective shape and material of the path, which made it necessary to throw away a large quantity of steel still available for crushing had the form been more suitable. There should be no difficulty in designing a better style of path, and in casting the sections of which it is composed of material of equal hardness.

The roll shells wore down very well, and showed no tendency to become seriously grooved or hollow in the centres. It was not necessary to face them up at all. In fact No. 2 when finishing pyritic ore to 300 mesh showed a perfectly fair surface. Certain small alterations in design would increase the life of the shells a little.

The belt conveyers and bucket elevator showed little signs of serious wear. The worm conveyer blades were originally made of too thin material, and did not last. This was easily remedied.

As the screening was frequently changed for experimental purposes, no calculation can be fairly made as to the cost of screens per ton crushed, but in normal work it should be less than in wet milling.

Power Required.—Mill.

To drive	Niagara pulveriser and screen	21	I.H.P.
„	No. 1 roll, reducing ½ in. to ¼ in. ...	20	„
„	No. 2 roll (finishing).....	23	„
„	Elevator.....	6	„
„	Screens	4	„
„	Screw conveyers	5	„
„	Belt conveyers	6	„
„	Electro magnet	1	„
	Total (friction included)	86	„

Two rockbreakers (No. 4 and No. 2) ...	26	
Ore lift to mill bins	11	
		— 37 I.H.P.

Dry Crushed Product.

The free milling (oxidised) rock (Table C) crushed to ⅛ and $\frac{1}{10}$ in., was easily screened when dry (moisture 1·1 per cent.), and 2·7 per cent. gave no serious trouble. When the ore was of the latter class there was a tendency to hang on the bottoms of the inclined shoots, &c. The passage of solutions through it was rapid. The degrees of fineness to which the pyritic ore was crushed can be seen in Tables D and E.

The 300-mesh product was slow in filtering. This was partly due to the depth of ore in the tank (7 ft.), and no doubt could be accelerated by a vacuum pump or by an ejector.

For *cost of milling*, see Table F.

Cyanide Treatment of Crushed Ore.

The tanks used were steel, 25 ft. diameter and 8 ft. deep, with usual filter bottoms and bottom discharge doors.

As it was not convenient to place the whole number in one line, although this would have been advantageous for filling, three tanks were put up close to the mill, and the remaining nine in a row some distance away, and a little below. The ore was treated a few days in the upper tanks and then transferred, a double treatment being thus obtained. The author does not consider double treatment to be of advantage in dealing with dry crushed material, as the ore is properly mixed in the first instance, and there is abundance of air entangled in the dry sand. He is rather of opinion that when the damp ore is transferred to the second row that there is a tendency to pack closer and to retard filtration.

Lime was added to the ore at the rock breaker floor, in proportions varying from 1 lb. to 2½ lb. per ton. By this means it was thoroughly mixed in the different machines, and any lumps were broken up.

The actual cyanide treatment differed a little from the ordinary procedure, and followed lines first suggested to the author by Mr. W. R. Feldtmann.

As soon as a tank in the first row was filled, a strong solution, 0·25 per cent. KCy, followed by two others, the last being 0·15 per cent., was pumped on to it. The time given to this treatment would be 66 to 70 hours. The solutions were not allowed to stand, but were drained off when the tank was once filled. The object of this was to allow fresh air to obtain access to the mass of damp sand. Each solution would be about 27 tons to the tank of ore (165 to 170 tons).

The ore was then transferred to the second row of tanks, having lost about 67 per cent. of its original assay value in this short time. It is probable that the abundance of air entangled in the dry sand, as compared with what would be held in a tank of sand settled with water, materially assisted the solution of the gold. Also the finer grains of the free gold would be quickly taken up.

A solution of 20 to 25 tons, not exceeding 0·20 per cent. KCy, was then pumped on to the transferred sands and drained off, the mass being allowed to remain damp for about 96 hours, when weaker solutions (0·15 per cent. to 0·10 per cent.) were used in continuous washes, making up a total of 75 tons per tank (2nd treatment). A water wash of 20 or 30 tons completed this part of the process, which would last some 275 hours.

A careful series of moisture tests and measurements of solution sumps during the treatment of four tanks (680 tons) showed that the total loss of liquid in the treatment was 96 tons, or 24 tons per tank. The moisture in the discharged residues averaged 12·3 per cent., or, say, 20 tons per tank, the remaining 4 tons would be represented by evaporation from surfaces of sumps and tanks and by leakages. As the ore contained about 3 tons of water per tank, in the form of moisture when delivered from the mill storage bin, the actual consumption of fresh liquid equals 21 tons per tank, or ⅛ ton (25 gallons) per ton of ore.

The solutions running from the first row of tanks carried from 13 dwt. to 32 dwt. per ton; those from the second row 3 dwt. to 4 dwt.; and the final wash 0·8 dwt. to 2½ dwt.

During the treatment of 22,500 tons, or say 130 tanks (representing solutions and washes equal to about 23,500 tons), the solutions

had been used over and over again, and it was considered advisable to ascertain if their efficiency was becoming weakened. The attention of the author had been attracted by a paper by Mr. Andrew F. Crosse, Past President of the Chemical and Metallurgical Society of South Africa, on "The Estimation of Oxygen in working Cyanide Solutions," read before that Society in August and September last. As it contained very valuable information for practical cyanide work, samples of the Luipaards Vlei solutions were sent to Mr. Crosse for examination. His report was substantially as follows:—

Sample A. *Mine water used in the cyanide works.*
1·33 miligrams oxygen per litre, or, say, $\frac{1}{5}$th of a saturated solution.

" B. *Strong cyanide solution.*
3·45 milligrams per litre.

" C. *Same solution as it commenced filtering off.*
No oxygen.

" D. *Same solution at end of filtering.*
2·8 milligrams of oxygen per litre.

" E. *Weak solution mechanically aërated.*
6·4 milligrams per litre or saturation.

" F. *Same solution running off tank.*
6·0 milligrams oxygen per litre.

The whole of the solutions in use were in good working condition.

Mr. Crosse's report on sample C shows that under certain conditions a solution may be deprived of its oxygen and be useless, and it is easy to imagine that this is often the case.

Sample E shows what can be effected by a very simple mechanical aëration in regenerating solutions.

The ore and solutions were tested at the same time for acidity and for lime. A sample was taken from a tank containing 160 tons ore to which 160 lb. lime had been added, and after treatment with water showed caustic lime equivalent to 33·6 lb. per 160 tons, or in excess of the amount required to neutralise acidity. The first solution coming off, however, carried lime equivalent to 36 lb. per 160 tons, or taken up in its passage. Practice showed that more than 1 lb. per ton was required during the whole treatment.

The ordinary zinc precipitation was found to be successful when sufficient lime had been added. Any deficiency caused a rise in the

solutions passing from the boxes, while too large an excess caused a whitish precipitate on the zinc, which was also detrimental. No difficulty was experienced in obtaining a good precipitation.

The remainder of the work in the cyanide department differs little from the usual practice where the gold slimes are treated by acid and filter press. The flux used in running down the bullion was, however, prepared from a formula given to the author by Mr. Crosse, and bullion has been obtained running 946 gold and 40 silver per 1,000, which shows what can be done in the way of obtaining clean gold if desired. The advantage does not, however, consist in the mere satisfaction of having clean bars; a distinctly better price is obtained for the gold, the slags contain less, and the cost of smelting decreases.

For *costs of cyanide treatment*, see Table G.

Extraction obtained.

The following figures give the results obtained to the end of October last, 25,133 tons having been treated:—

		Oz.
Average assay value per ton (2,000 lb.), 8·28 dwt.		
Total contents, in fine gold		10,424·17
Residues, average assay value, 1·4 dwt.		
Contents, in fine gold		1,758·49
Balance, theoretical extraction, 83·13 per cent.		8,665·68
	Oz.	
Fine gold banked, bars $\frac{1}{19}$	8,275·87	
„ sold, by products	128·22	
„ recovered in November	47·73	
		8,441·82
Balance to be accounted for		213·86
	Oz.	
Fine gold in slags	25·24	
„ „ zinc extractor boxes	28·27	
„ „ sump solutions	40·53	
„ unaccounted for, 1·15 per cent.	119·82	
		213·86

It is very probable that the zinc boxes contain more gold than has been credited. Also, flue scrapings, ashes, crucibles, &c., will all contribute to decrease the unaccounted for.

If the extraction were worked out, as is usually the case on the Rand, that is on the total weight of gold won plus gold (by assay)

in residues and slimes, the value of the ore would be decreased to 8·199 dwt., and the actual extraction raised to 82·74 per cent.

Coarse Crushing, Free Milling Ore.

The following notes of a test crushing made at Luipaards Vlei may be of interest to some.

Nine hundred and seventeen tons of oxidised "banket" were roughly crushed in a small Gates breaker (No. 1); about $\frac{1}{8}$th of the whole quantity would not pass through a $\frac{1}{2}$-inch mesh. General details are given in Table H.

The plant used was a makeshift arrangement, but nevertheless 68·9 per cent. of the gold was recovered or accounted for.

The last 400 tons treated, although not crushed so well, gave an extraction of 73 per cent., the improvement being due to experience gained in the first 517 tons. Longer time was given, and lime (3 lb. per ton) was added. The ore also remained damp, not covered by solution, for a longer period.

The coarser portions after cyaniding, when crushed fine and panned, showed gold which proves that some of the value of the residue must be attributed to undissolved gold which the cyanide had not reached.

A small portion of the residue value may consist of dissolved gold remaining in the small cavities in the cement, or in the oxidised pyrites, and which had not been leached out by the water washes.

The residues would be worth about 11*s.* 4*d.* per ton. A very slight expenditure of power, and simple machinery would reduce the whole to, say, $\frac{1}{8}$ inch, and increase the extraction to 85 per cent., bringing down the residue value 3*s.* 7*d.*, at a cost which would not exceed 1*s.*

In conclusion, the author would plead indulgence for any discrepancies which may be noted in percentages of weight, assay values, &c., in the tables accompanying this paper.

He has found that unless special care is taken, at a cost which is often economically prohibitive, it is not possible to take two large samples from the same charge before and after treatment, which will absolutely agree in sizes or in value, and therefore prefers to give the results as they worked out rather than to "adjust" them. Their practical value will be as great as if they agreed absolutely.

TABLE A.

Work done by Crushers.

Niagara Pulveriser.

Free milling ore.			Pyritic ore.		
Size of mesh.	Ore.		Size of mesh.	Ore.	
	Entering.	Leaving.		Entering.	Leaving.
in.	per cent.	per cent.	in.	per cent.	per cent.
+ 3/8	51·2	0·00	+ 3/8	93·5	0·00
+ 1/4	29·3	64·30	+ 1/4	6·5	80·5
+ 1/8	3·7	7·70	+ 1/8	—	6·7
+ 1/16	6·7	11·80 }	+ 1/16	—	6·7
− 1/16	9·1	16·20 }	+ 1/20	—	1·3 }
			− 1/20	—	4·8 }

Roll No. 1. Coarse.

Free milling ore.			Pyritic ore, 1/14 in.		
Size of mesh.	Ore.		Size of mesh.	Ore.	
	Entering.	Leaving.		Entering.	Leaving.
in.	per cent.	per cent.	in.	per cent.	per cent.
+ 1/4	72·7	19·6	+ 1/4	60·4	10·5
+ 1/8	6·9	9·4	+ 1/8	2·6	10·5
+ 1/16	7·9	29·4 }	+ 1/10	15·5	31·2
− 1/16	12·5	44·6 }	+ 1/20	12·9	18·7 }
			− 1/20	8·6	29·1 }

Roll No. 2. Finishing.

Free milling ore.			Pyritic ore, 1/14 in.		
Size of mesh.	Ore.		Size of mesh.	Ore.	
	Entering.	Leaving.		Entering.	Leaving.
in.	per cent.	per cent.	in.	per cent.	per cent.
+ 1/8	32·0	0·4	+ 1/8	22·5	0·9
+ 1/16	42·7	47·3 }	+ 1/16	40·0	21·3
− 1/16	25·3	52·3 }	+ 1/20	27·5	37·7 }
			− 1/20	10·0	40·1 }

Note.—The bracketed quantities represent finished product.

TABLE B.

Wear of Shells of Rolls and Pulveriser.

***Ore Crushed*, 22,533 *Tons*. *Average Milling Time*, 72 *days*.**

	Pulveriser.	Two rolls.	Totals.
	lbs.	lbs.	lbs.
Weight of material worn.....	2746	7325	10071
„ „ thrown out	4808	3281	8089
Weight of path and roll	7554	—	—
„ shells.....	..	10606	18160
Cost of material per ton	2·258*d.*	3·536*d.*	5·794*d.*

TABLE C.

Comparison of Charge and Residue (Sieving Test).

Charge—Free Milling Ore ¼ in. Mesh.

Mesh.	Weight, per cent.	Assay, dwt.	Dwt. in 100 tons.
in.			
+ $\frac{1}{20}$	34·8	5·7	198·36
+ $\frac{1}{60}$	35·1	7·3	256·23
− $\frac{1}{60}$	30·1	15·0	466·55
			921·14 or 9·21 dwt. per ton.

Residue*—1·29 *dwts.

Mesh.	Weight, per cent.	Assay, dwt.	Dwt. in 100 tons.	Extraction, per cent.
in.				
+ $\frac{1}{20}$	34·5	1·7	58·65	70·4
+ $\frac{1}{60}$	38·5	1·2	46·20	81·9
− $\frac{1}{60}$	27·0	0·9	24·30	94·8
			129·15 = 1·29	86·0

Table D.

Panning Test. Charge and Residue. Pyritic Ore, crushed to 400 mesh.

Charge.—Assay value, 8·76 dwt.

Classification.	Per cent. weight.	Assay, dwt.	Dwt. in 100 tons.	Per cent. of contents.
Coarse stuff + $\frac{1}{24}$ in.	5·01	2·500	12·525	1·43
Sands	74·81	3·833	286·823	32·74
Slimes	16·00	3·208	51·328	5·86
Fine concentrates	1·92	90·000	172·800	19·73
Coarse concentrates	2·26	83·000	187·580	21·40
			711·056	81·16
Free gold, say	..	..	165·650	18·84

Residue.—Assay value, 0·812 dwt. 18¼ days.

Classification.	Per cent. weight.	Assay, dwt.	Dwt. in 100 tons.	Extraction, per cent.
Coarse stuff + $\frac{1}{24}$ in.	4·52	1·4166	6·4030	48·87
Sands	75·50	0·2916	22·0158	92·35
Slimes	16·46	0·1250	2·0575	95·98
Fine concentrates	1·40	10·500	14·7000	92·00
Coarse concentrates	2·12	17·000	36·0400	80·70
			81·2163	90·80

Note.—Extraction was probably completed in 14 days. As only 71 tons were treated, the ore was only about 3 ft. deep in the tank. Moisture in residues 12·3 per cent. Weight, 1 cubic ft. dry sand, 87 lb.

Sieving Test of same product.

+ $\frac{1}{30}$-in. mesh	29	per cent.
+ $\frac{1}{60}$,, ,,	35	,,
+ $\frac{1}{90}$,, ,,	14	,,
− $\frac{1}{90}$,, ,,	22	,,

TABLE E.

Sieving Test. Charge and Residue.—Pyritic Ore. 300 *Mesh.*

Charge.—Assay Value, 6·5 *dwt.*

Mesh.	Per cent. weight.	Assay, dwt.	Dwt. in 100 tons.	Per cent. of contents.
in.				
+ 1/30	2·4	4·0	9·60	1·3
+ 1/50	20·9	3·5	73·15	10·9
+ 1/60	37·2	6·5	241·80	35·7
+ 1/80	12·8	8·5	108·80	16·0
− 1/80	26·7	9·2	245·64	36·1
		6·789	678·99	

Residue after 14 *days. Assay* 1·5 *dwt.*

Mesh.	Per cent. weight.	Assay, dwt.	Dwt. in 100 tons.	Per cent. of contents.	Per cent. of extraction.
in.					
+ 1/30	2·9	1·25	3·625	2·4	62·26
+ 1/50	23·3	1·75	40·775	28·2	44·30
+ 1/60	34·2	1·70	58·140	40·3	75·96
+ 1/80	12·4	1·20	14·880	10 3	86·33
− 1/80	27·2	1·00	27·200	18·8	92·33
			144·62		77·00

Note.—The ore probably assayed more than 6·5 dwt.

TABLE F.

Cost of Milling Pyritic Ore.

		s.	*d.*
White wages	per ton	0	10·995
Native ,,	,,	0	3·924
Stores	,,	0	8·863
Lighting	,,	0	0·485
Steam power	,,	0	7·856
Electric power	,,	0	0·408
Maintenance, &c.	,,	0	7·322
Total		3	3·853
Rock breaker cost			9·747

Table G.

Cost of Cyaniding.

		s.	d.
White wages	per ton	0	11·18
Native „	„	0	7·20
Maintenance	„	0	3·45
Assay charges	„	0	3·60
Cyanide 0·40 lb.	„	0	7·70
Zinc 0·31 lb.	„	0	1·08
Lime 2·55 lb.	„	0	1·15
Light and Power		0	2·30
Stables		0	0·85
Coal, coke, and sundries		0	1·00
Total		3	5·86

Summary of Caynide Cost.

		s.	d.
Tanks and Extractor house work	per ton	2	3·30
Clean up and smelting, &c.	„	0	5·15
Discharging residues	„	0	8·36
Sundries		0	1·05
Total		3	5·86

Table H.

Trial crushing (coarse) Free Milling Ore.

	Dwt.		Contents, oz.
Charges—			
917 tons, average assay	9·625	fine gold	441·356
Residues—			
916 tons, average assay	2·708	„	124·245
Theoretical extraction	6·917		317·111

or 71·84 per cent.

Recovery—	Oz. fine.
Bullion, 374·3 oz.	290·3
Slags and sumps	13·6
	303·9

or 68·91 per cent. actual.

Analysis of Residues.

	Weight per cent.	Extraction per cent.
Material remaining on $\frac{1}{2}$-in. mesh ...	19·70	56·60
" passing $\frac{1}{2}$-in. + $\frac{3}{8}$-in. " ...	17·37	65·17
" " $\frac{3}{8}$-in. + $\frac{1}{8}$-in. " ...	19·80	66·00
" " $\frac{1}{8}$-in. + $\frac{1}{16}$-in. " ...	23·70	77·60
" " $\frac{1}{16}$-in. mesh	19·00	81·00
Total.....		69·31

DISCUSSION.

Mr. A. G. Charleton thought that Mr. White had presented the Institution with a most valuable and instructive paper, and considered that the facts the author had laid before them pointed to the probability that direct cyanide treatment preceded by dry crushing would play a much more important part in metallurgical practice than might have been anticipated a few years ago.

Seeing that in the case in question it appeared possible to treat the whole of the ore in one operation, and extract a reasonably large percentage of gold without having recourse to amalgamation and separate treatment of the slimes, it ought to be able to compete successfully in many instances with wet crushing. In considering the adaptability of direct treatment for any special ore, it was evident that the surrounding circumstances had to be carefully weighed in forming an opinion upon the probable commercial results which the method would give, as it was liable to be affected by various side issues. Very divergent opinions as to the scope of dry crushing were formerly held by leading metallurgists, and Dr Hatch, in *The Gold Mines of the Rand*, quoted the views of Messrs. A. F. Crosse, W. S. Hamilton, Butters, Bettel, Williams, and Von Gernet, which were well worth careful perusal by anyone who contemplated the adoption of the direct method of treatment. The conclusions of the last four authorities named were certainly not altogether favourable, and if it is the case, as Mr. Butters says, that "Before the close of 1896 the majority of all the wet-crushing mills on the Witwatersrand will be obtaining over 90 per cent. of the gold at a cost not exceeding 7*s.* a ton for metallurgical operations dealing with 15-dwt. ore," it would appear that the results at the Luipaard's Vlei Estate, showing a maximum extraction of 82·74 per cent. at a cost of 7*s.* 7·52*d.*, afford some justification for questioning the advantages

of dry crushing. Still there were certain districts, such as Kalgoorlie, in Western Australia, where dry crushing and cyaniding (treating the sands and slimes, whether together or apart) was bound to take the place of wet crushing to a great extent owing to special local conditions. Whilst, therefore, he agreed with Mr. White in regard to the adaptability of dry crushing to certain kinds of ore, he did not think that the advantages of crushing with rolls as compared with stamps were by any means proved, as the author appeared to claim in his paper.

In Mr. Collins's Presidential Address in March, 1895, he put the cost of crushing with rolls to 30-mesh as being between 1*s.* 6*d.* and 3*s.* 6*d.* per ton, whilst stamping with ordinary stamps to 30-mesh he estimated cost about 1*s.* to 1*s.* 9*d.* per ton. The figures he gave for breaking hard ores to road metal size were 2*d.* to 4*d.* per ton, and for grinding hard rock, already reduced to $\frac{1}{16}$ in. or less, in iron mills to 50-mesh, he estimated at 1*s.* to 1*s.* 6*d.*

It was well known that in many cases where free gold-ores had to be dealt with, it was possible to extract 70 to 94 per cent. of their gold contents by wet stamping and amalgamation for 1*s.* 5*d.* to 2*s.* a ton, and even ores which contained a considerable percentage of pyrites (yielding a concentrate which required to be smelted or chlorinated), could be treated at a cost of 1*s.* 6*d.* to 3*s.* 11*d.* per ton, giving an extraction of 92 to 94 per cent.

Mr. T. A. Rickard had given the cost of milling in the Black Hills as 2*s.* 11*d.*; at Grass Valley, Cal., 3*s.* 4½*d.*; and in Amador County 1*s.* 10½*d.*; whilst the cost at the Alaska Treadwell in 1893 was only 1*s.* 5½*d.* On the Rand the milling costs at the Meyer and Charlton and Glencairn, crushing with 50 stamps, was given by Dr. Hatch as 3*s.* 5·79*d.* and 3*s.* 7*d.* per ton respectively, inclusive of concentration. He chose the latter mills because, taking the duty of stamps weighing 850 to 950 lb. at an average rate of 4 tons per head per diem, the two batteries would have about the same capacity as the plant described in the author's paper, the cost of crushing in which was stated to be 4*s.* 1·6*d.* per ton.

That wet stamps were cheaper crushers than rolls, could be demonstrated in several ways. For instance, it might be safely assumed that a battery of 900 lb. stamps with 96 drops a minute falling 8 in. would easily pass 4 tons of "banket" per head through 700 or 800-mesh screens, with an expenditure of not more than 150 I.H.P., including preliminary crushing with rock breakers. This would show an efficiency of about 1 ton 6 cwt. per I.H.P. per 24 hours as compared with 1 ton 12 cwt. crushed in the roll mill, representing approximately 1·3 cwt. per I.H.P. per hour, a result which agrees

with a statement made by Professor H. Louis, who puts the effective power of a Krom roll at 1·04 cwt. per I.H.P. per hour. It had to be borne in mind, however, that whilst the rolls were only crushing to 300-mesh, stamps crushing to 700-mesh were producing a product more than double as fine; in other words, it seemed fair to conclude that crushing to the same degree of fineness, a roll plant would require almost double the power of stamps to put through the same quantity of stone, thereby doubling the cost of fuel required to furnish the requisite power to run such a plant for a specified time. In proof of this it might be observed that, running 60 to 70 heads, the Robinson Company, with 930-lb. stamps, crushing 4·6 tons per diem through 900-mesh screens, did so at a cost of 8·24*d.* per ton, whilst Mr. White's paper indicated that crushing to only 300-mesh the roll mill cost 7·856*d.* for steam power per ton, exclusive of electric light.

He considered that the cost of such items as labour, stores, and maintenance were distinctly in favour of stamps as compared with rolls, as would be seen from a table which he had prepared:—

Robinson Mill, 1894.	*d.*	*Luipaard's Vlei, 1898.*	*d.*	
Rock breakers	4·97	Bock breaker	9·747	per ton.
Mill labour	6·42	Labour { White	10·995	„
		Labour { Native	3·924	„
Supplies	7·33	Supplies	8·863	„
Fuel	8·24	Fuel, including light	8·264	„
Water	3·89	Lighting	0·485	„
Maintenance and repairs	4·20	Maintenance	7·322	„
	2*s.* 11·05*d.*		4*s.* 1·600*d.*	„

Mr. White had not stated what the erection of the Luipaard's Vlei plant will cost, but he imagined that it was more than that of a stamp mill of similar capacity delivering the same size product, in which case the working costs would be further increased by the sum set aside for depreciation of machinery and redemption charges. Had it been necessary to crush the ore to the same degree of fineness as in the stamp mill, he thought that the Luipaard's Vlei costs would have been nearly doubled, as it would have taken about twice the time to put the same amount of stone through the roll mill, or else the plant would have to be duplicated.

He had considerable experience with rolls, and for special purposes (such as in concentrating works) where the ore requires to be gradually reduced in size, and where it was advisable to obtain an evenly graded material of coarse, sharp, sand-size (1 mm.

upwards) with a minimum production of "slimes," he considered they were the very best machines to employ; but the figures given in Mr. White's paper rather strengthened the opinion he held, (1) that rolls were unsuited to deal with even a concentrating-ore if the material was finely disseminated, as in Cornish "capel" and some lead ores; (2) that they were ill-adapted for any ore which must be broken in one operation from lump form to comparatively fine size; (3) that they were also out of place when amalgamation of any kind was to follow directly after crushing, especially when large quantities of ore carrying finely disseminated free gold had to be handled wet. He also thought that clayey ore in a damp condition could not be dealt with satisfactorily with machinery of the roll class.

To sum up, he considered the use of rolls was the weak point in the author's suggested method of treatment, as the figures given showed that their use involved the larger part of the cost of the process, and he believed that it was highly probable that a recent improvement (to which he alluded at one of their last meetings) would enable dry stamps to successfully compete in cost with rolls in the matter of reducing ore for cyanide or other treatment, dry, to a given degree of fineness. Mr. A. F. Crosse, in reporting on an experimental plant erected at the Village Main Reef, said that he believed the process would prove a success if it received a fair trial under proper management. He did not see why the efficiency of stamps should be less with *air* in the mortar boxes instead of water, providing the crushed material could be removed rapidly, indeed it ought certainly theoretically to be greater, and the inventor now claimed to have overcome the chief objection which was at first experienced with the dust.

After all, what chiefly concerns a company is to obtain a maximum extraction at a minimum cost, and to effect this the machinery and method of treatment must be well considered in adapting them to the ore.

Mr. Alfred James thought that Mr. White was very modest in calling the crushing of 22,000 tons of ore an experiment. The great point with regard to dry and wet crushing was that of output. He spoke on the subject before the Chemical and Metallurgical Society at Johannesburg in October, 1895, and enlarged on all the advantages of the dry crushing which were then being reaped in New Zealand and elsewhere, but stated that as long as they had on the Rand an output of 5 tons crushed per head per day, it was practically out of the question to go into the dry crushing problem until that process could show an output which would be very nearly equal

to the enormous output on the Rand. He suggested that rolls might obtain that result, and said he would like to see them tried. Mr. White had solved the question of output; if they took the figures given by Mr. White, it would be found that he was making an output of over 2 tons per head per day per horse power used, whereas the best practice he knew on the Rand came to about 1½ ton per head per day. He thought they ought to offer Mr. White their congratulations on producing a greater output by the dry process than was obtained by the wet. Having obtained the output, there were two other factors to be considered, viz., the cost of obtaining it, and the extraction. With regard to the cost of dry crushing, he did not think they had sufficient data, although Mr. Charleton had given them certain costs for a mill running continuously for 24 hours, and it was a question whether a comparison of costs could be made between a mill running full time and one working at odd times. With regard to cyanide consumption, it would naturally be expected that in treating an ore direct which was richer than the average tailings on the Rand, there would be a heavier consumption of cyanide, yet Mr. White's consumption was one of the lowest on the Rand.

With regard to extraction, anyone could work up to high extraction, but Mr. White had naturally chosen to go in for what he considered the highest profit on such low grade ore, but personally he thought there should have been higher extractions from the material. Looking at the figures, he noticed that where the fines were separated from the coarse the author appeared to have no difficulty in getting an extraction of from 92 to 96 per cent.; it was the coarse particles which had lowered the extraction, and he would be glad to hear from Mr. White why he did not treat the coarse particles separately or re-crush them.

Mr. White referred to double treatment, and for dry crushed ore one was inclined to agree with him. It had to be remembered that in leaching a vat (especially a deep one) the last 2 or 3 ft. did not seem to have the same chance of extraction as the rest of the material. By having double treatment the lowest layer got on the top of the next vat, and there was a chance of obtaining a higher extraction at very little extra cost. On the other hand, there was the question of packing, and he thought that Mr. White's experience in that respect was due to the fact that he dumped his finely crushed ore from one vat to another without its being sufficiently dry, whereas if a vacuum had been used to draw off the greater portion of the water the result would have been better.

He did not advocate universal dry crushing, but was of opinion

that it was a factor to be seriously considered. With regard to its adoption in South Africa, if they took the *State Mining Engineer's Report* for 1897, they would find that during the last quarter of that year no less than 33,000 tons of ore were dry crushed. One great essential to success was that the plant should be thoroughly adapted to the ore to be treated. He knew of a dry crushing plant in the Transvaal which was entirely different from that which Mr. White was using, and which paid very good dividends, but that was on different type of material. Mr. White had shown some exceedingly good results, and he considered that had it not been for the skill and care exercised in regard to the selection of his plant for the precise material to be treated, he would not have been able to place such a good statement before them.

The President announced that, owing to the lateness of the hour, the further discussion of Mr. White's paper would be postponed till the next meeting.

FIFTH ORDINARY MEETING, 15th February, 1899.

Mr. A. C. CLAUDET (*Hon. Treasurer*) in the Chair.

The Minutes of the previous meeting were read and confirmed.

"Notes on Economical Mining at the Myalls United Gold Mine, New South Wales."

By Wm. R. Thomas, Assoc. Inst. M.M.

For the purpose of comparison with results obtained in other parts of the world, the following notes, the figures of which are compiled from the working cost statements of the Myalls United Gold Mine Company, Limited, will possibly be of interest.

The general manager at the mines is Mr. Thomas White, under whose supervision the cost sheets are prepared. The author, having lately spent four months professionally at the mine, and being connected with its London management, is in a position not only to vouch for their accuracy, but at the same time is enabled to give a fairly accurate description of the surrounding conditions which naturally affect the cost of working to a greater or less extent.

In order to avoid any accusation of overstating results, and at the same time to keep the length of the paper within reasonable limits, it is proposed to touch, although briefly only, upon the situation of the mine, the facilities which exist for economical working, the nature of the lodes, and the cost and character of the plant, besides giving some details as to how the working costs are arrived at.

The mine is situated in the Tomingley district, 35 miles from the town of Dubbo, on the Western Railway, which is 300 miles distant from Sydney. The surrounding country is perfectly flat, and the cart road from Dubbo to the mines is a fairly good one.

Two important reefs have been proved to exist in andesite, which is partially decomposed near the surface, but in depth gradually

becomes harder—irregular and excessively hard bars of a similar rock running through the formation.

The reefs are known as the "Main Reef" and "Queen Reef"; they are approximately parallel, and are separated by 63 ft. of country rock. The development of the Queen Reef has thus far proved it to be persistent in depth as well as along its "course." It has a bearing of 15° W. of N., and dips to the east at an angle of 75°. The developments now extend for a distance of 500 to 600 ft. along its course, and the lowest level has been opened at a vertical depth of 336 ft.

The reef varies in width from 2 to 20 ft., but its average width may be taken at 8 to 9 ft., and it consists of massive quartz carrying a small percentage of mineral. The cost of drivages varies from £1 to £1 10*s.* per ft., which does not include either tramming or hoisting.

The Main Reef, to the depth of No. 3 level, varies in width from 2 to 15 ft., and is composed of quartz of a similar nature to that produced by the Queen Reef.

The main shaft is vertical, and has been sunk to a depth of 350 ft. Crosscuts intersect the lodes at the respective depths—136 ft., 236 ft., and 336 ft. The shaft, 12 ft. by 4 ft. within, is close timbered, and is divided into three compartments having two cageways and a ladder and pumping compartment. It is located so as to intersect the lode at a depth of from 500 to 600 ft., which will secure an economical system of working to a depth of 1,000 or more feet.

(*Note.*—Up to the present the pumping charges are practically nil.)

The cages are fitted with Cowling's patent grippers, and safety hooks are provided for prevention of overwinding.

The trucks have a capacity of 7 cwt., and sufficient quartz is hoisted in one 8-hour shift to keep the mill at work full time.

The plant consists of a 40-stamp mill (850 lb.), colonial make, driven direct by a compound engine of trunk guide type, of sufficient power to drive 60 heads, and the battery is supplied with amalgamating tables, blanket strakes, two Berdan pans, White's improved Denny pan for grinding concentrates, together with the usual accessories for securing up-to-date amalgamation and battery extraction.

The hoisting engine (duplex), together with compound air-compressor of 5-drill capacity and rock-breaker engine, are adapted to work expansively by the use of a surface condenser. The boilers (steel) are high pressure, and of the Lancashire type, fitted with Galloway tubes.

The cyanide plant consists of eight percolating or leaching vats of 40 tons capacity, three solution sumps, and one elevated or store sump, also four settling vats. The former 12 vats are built of wood, with round iron bands, fitted with tightening screws. There are four zinc boxes, one pump for circulating the liquor, assay office, roasting and smelting rooms; also an elevator driven by a small stationary engine. The capacity of this plant is equal to 2,200 tons per month of 28 days.

The cost of the mining plant as described, excepting the cyanide plant, is as follows:—

	£	s.	d.
Machinery, and all supplies in connection therewith	7,429	4	10
Carriage, rail freight, &c., on same	1,247	19	8
Labour employed in erection	992	8	1
	9,669	12	7
Additional supplies (extras)	317	8	7
Total cost	9,987	1	2

The cost of the cyanide plant complete as it now stands (2,200 tons capacity), including purchase, freight, and labour in erection, comes to £2,088 7*s.* 4*d.*

The only permanent supply of water in the district available for milling and other purposes is found at a creek at least 6½ miles from the mine. A dam was constructed at this creek securing a conservation of water, and the water is now conveyed to the mine through a line of light galvanized steel pipes made by Zollner & Co., of Sydney. The difference of levels between these two points is just sufficient to allow of a flow of water by gravitation.

The initial cost of this undertaking amounted to £3,614 6*s.* 2*d.*

The cost of laying open the mine to secure the monthly output—including main and other shafts, crosscuts, drivages, &c., at the end of 1897, amounted to £8,464 16*s.* 2*d.*, which sum is now gradually being written off.

The total cost therefore in laying out this mine for an economical and effective system of working, as per Company's Statement of Accounts to end of 1897, has been as follows:—

	£	s.	d.
Total cost of machinery and its erection	9,987	1	2
Cyanide plant, 1897–98	972	10	11
	1,115	16	5
Water supply	3,614	6	2
Mine development account	8,464	16	2
Buildings account	2,259	18	11
	26,414	9	9

The ore, after passing through a Blake rock breaker, is fed into the mortar boxes by Challenge ore feeders, and milled through punched screens containing 200 holes to the sq. in., equal to about 30-mesh wire screens. The duty varies from 2¼ to 2½ tons per stamp per 24 hours.

The tailings, after passing over the plates, are elevated to settling tanks at a height of 30 or more feet by two 12-in. plunger pumps, driven from the cam shaft by means of rope gearing. From the "settling tanks" they are run into the leaching vats, and after cyanidation are used for filling the mine stopes.

From an ore having a value of 6 dwt. we are able to extract 4 dwt. by amalgamation, 1½ dwt. (fine gold) by cyanidation, the residual tailings having an assay value of 10 to 12 gr. per ton.

An increase in the value of the ore gives a corresponding increase in percentage of extraction—for instance, the average value of ore treated during the month of September was nearly 8 dwt., and the assay value of the residual tailings was 11 gr. per ton.

The ore, consisting of practically clean quartz, gives only a small percentage of slimes, which up to the present have not been treated, and which it may be said have not been taken into account in the foregoing extraction results.

A General Monthly Statement is prepared, giving the total cost, as well as cost per ton, under the following heads :—

(*a*) *Mining*, covering all costs entailed in mine development, stoping, timbering, mullocking, tramming, &c., together with all underground supervision ; and all supplies and timber.

(*b*) *Surface*, including blacksmiths, carpenters, engineers, timekeeper, and storekeeper, deckmen, &c.

(*c*) *Milling*, including rock-breaking, elevating, filling of hoppers, supervision of mill, amalgamating, engine power, wear and tear of shoes and dies, screens, liners, and loss of quicksilver, driving of Berdan and other pans, elevating tailings, &c.

(*d*) *Cyanidation*, covering every cost connected with the cyaniding

of tailings as received from the mill, and placing the residues on the waste tailings dump, assaying and smelting of bullion.

(*e*) *Administration, &c.*, covering cost of management, local secretary (legal manager), office and salary, rents, insurances, postage, cables, &c.

It is also well to note the following prices as paid by the Company:—Labour, 7*s.* 6*d.* to 9*s.* per day; dynamite (gelignite), 1*s.* 3*d.* per lb.; cyanide, about 1*s.* 2*d.* per lb.; fuel 7*s.* per cord, candles 5*d.* per lb.

The following are the costs per ton for the last six months.

TABLE No. 1.—*Monthly Working Costs.*

Date.	Department.	Total Cost.			Cost per ton.		Remarks.	
1898.		£	s.	d.	s.	d.		tons.
July 9	Mining	738	19	7	6	8·25	Mined	2210
	Surface	191	17	8	1	8·83	Milled	2171
	Milling	227	10	3	2	1·1	Cyanided	1950
	Cyaniding	336	14	9	3	5·4		
	Administration charges	115	3	6	1	0·7	Month's output ...	2171
		1610	5	9	15	0		
Aug. 6	Mining	746	5	11	6	7·8	Mined	2244
	Surface	158	16	5	1	4·9	Milled	2209
	Milling	206	14	10	1	10·4	Cyanided	2217
	Cyaniding	348	4	10	3	1·6		
	Administration charges	115	3	6	1	0·5	Month's output ...	2209
		1575	5	6	14	1		
Sept. 3	Mining	705	10	3	6	3·1	Mined	2253
	Surface	169	5	10	1	6	Milled	2153
	Milling	198	14	0	1	10·1	Cyanided	2062
	Cyaniding	343	2	0	3	3·9		
	Administration charges	115	3	6	1	0·8	Month's output ...	2153
		1531	15	7	14	0		

Explanatory Note.—The cost per ton in the foregoing five departments is arrived at by dividing the total cost in each department by the number of tons landed or treated in that particular department.

TABLE No. 1.—*Monthly Working Costs—continued.*

Date.	Department.	Total Cost.			Cost per ton.		Remarks.	
1898.		£	s.	d.	s.	d.		tons.
Oct. 1	Mining	648	9	1	5	10·1	Mined.....	2220
	Surface	168	9	9	1	6·2	Milled	2185
	Milling	197	11	8	1	9·7	Cyanided	2200
	Cyaniding	334	8	1	3	0·48		
	Administration charges	115	3	6	1	0·65	Month's output ...	2185
		1464	2	1	13	3		
Oct. 29	Mining	720	1	4	6	1·79	Mined.....	2342
	Surface	197	13	3	1	8·25	Milled.....	2342
	Milling	193	0	1	1	7·77	Cyanided	2250
	Cyaniding	357	2	2	3	2·09		
	Administration charges	115	3	6	0	11·80	Month's output.. ..	2342
		1583	0	4	13	8		
Nov. 26	Mining	743	2	9	6	0·56	Mined.....	2458
	Surface	176	4	8	1	5·20	Milled.....	2406
	Milling	213	8	9	1	9·34	Cyanided.....	2400
	Cyaniding.. ...	356	19	11	2	11·70		
	Administration charges	115	3	6	0	11·48	Month's output....	2406
		1604	19	7	13	2·28		

TABLE No. 2.—*Statement of Mine Development for the year* 1898.

Month ending.	Queen Reef.				Main Lode.			Grand total develop-ment.	Cost per ton on mine output.	Amount of reserves.
	Driving.	Rising and sinking.	Cross-cutting.	Total.	Driving.	Rising and sinking.	Total.			
	ft. in.	ft.	ft.	ft. in.	ft.	ft.	ft.	ft. in.	s. d.	tons.
January 22	23 0	5	..	28 0	..	14	14	42 0	1 0	22,098
February 19.....	138 6	30	..	168 6	..	..	..	168 6	2 9	22,290
March 19	128 6	32	22	182 6	..	32	32	214 6	3 4	24,024
April 16	108 0	17	30	155 0	10	45	55	210 0	2 5	26,099
May 14	70 0	18	24	112 0	23	..	23	135 0	1 8	25,884
June 11	38 0	75	37	150 0	7	4	11	161 0	1 9	25,438
July 9.....	103 0	59	54	216 0	..	22	22	238 0	1 11	25,178
August 6	112 0	117	20	249 0	..	..	..	249 0	2 0	25,234
September 3	91 0	8	15	114 0	47	20	67	181 0	1 6	25,181
October 1	45 0	6	35	86 0	67	6	73	159 0	1 11	25,311
,, 29.....	47 0	44	..	91 0	21	26	47	138 0	1 7	25,194
November 26....	130 0	40	..	170 0	19	27	46	216 0	1 6	25,386
December 24....	88 0	32	9	129 0	..	76	76	205 0	1 10	25,335
Total.....	1122 0	483	246	1851 0	194	272	466	2317 0	±2 0	±25,335

TABLE No. 3.—*Analysis of the Average Costs for six months from 9th July to 26th November*, 1898, *as per Table No.* 1.

	Cost per ton.		Total cost per ton.	
	s.	*d.*	*s.*	*d.*
1. *Mining.*				
Developing and stoping	4	1·12		
Tramming	0	8·04		
Mullocking	0	6·02		
Timbering	0	5·67		
Supervision (underground)	0	6·38		
			6	3·23
2. *Surface.*				
Landing	0	1·77		
Winding and air compressing ..	0	4·54		
Blacksmiths	0	4·59		
Store-keeper, freight, repairs, and general expenses	0	8·02		
			1	6·92
3. *Milling.*				
Labour and supervision	1	0·74		
Supplies and repairs	0	4·88		
Fuel	0	4·05		
			1	9·67
4. *Cyaniding.*				
Labour, including supervision ..	1	5·13		
Stores consumed	1	8·52		
Fuel	0	0·44		
			3	2·09
5. *Administration*			1	0·31
Total			13	10·22

The following is a fair analysis of the cyanide costs—

	s.	*d.*
Labour (filling vats)	1	0·3
Supervision	0	3·7
Boys	0	1·0
Labourers (extra)	0	1·0
Tip-timber	0	0·1
Steam	0	1·2
Lime	0	0·5
Coke	0	0·6
Chemicals	0	0·5
Zinc	0	1·7
Cyanide	1	1·7
Sundries	0	0·5
	3	0·8

The foregoing figures do not provide for depreciation of plant and machinery (other than wear and tear), or London management and offices; but it can be fairly taken that a charge of about 2*s.* per ton would cover this.

Note.—Bank charges on sale of bullion amount to about 4*d.* per oz., which are not rightly chargeable against working cost, and, therefore, are not included.

During the six months in question the mine developments have been practically equal to the output. The reserves are about equal to one year's output for the present mill capacity. (See Mine Development Account.)

The author tenders his thanks to the directors of the Myalls Company for granting the necessary permission to publish these working costs in this manner, and trusts that the information contained in these few notes may be of interest to the older members, and, possibly, of some value to the younger members of this Society, who may in the future become connected with similar low-grade propositions in our colonies or other parts of the world.

There can be no doubt that low-grade schemes must, if successful, be furnished with the most modern and complete plants, and be extensively worked in a business-like manner in every way.

DISCUSSION.

The Chairman thought that the information as to costs which the author had given in such detail was most valuable. At the property in question a 6-dwt. ore was being treated at a considerable profit, which was, in his opinion, only another instance of how cheaply such work was carried out in the colonies, as compared with some other parts of the world.

Mr. J. H. Collins agreed with the Chairman as to the usefulness of the paper; in fact he considered it in some respects a model one, while the comparisons of costs were of very great value. Although it was not possible to make exact comparisons between different mines, he was reminded of the results obtained at Dolcoath, full particulars of the last three years' work there being now available. Dolcoath turned out more ore than Myalls, and in that respect a little economy might be expected. On the other hand, however, it was a very much deeper mine, the ore was very much harder, and he thought they would all agree that it was a more costly process to get tin out of a tin ore than gold out of a

gold ore. Allowing for such differences, he thought they need not go abroad to see good work. During the last six months 40,000 tons of ore had been dealt with at Dolcoath, at a gross cost of £1 0*s.* 3*d.* per ton, as against 14*s.* at Myalls. If they considered that it cost 3*s.* a ton for pumping, and that the amount stated included 1*s.* a ton for dues at Dolcoath, and that the ore was much harder, he thought that the home mine did not compare badly with the foreign one. Taking all the circumstances into account, there was no doubt that in both cases a great deal of work was being done for a very reasonable amount of money. The average depth at which the ore was being worked at Dolcoath was not less than 1,800 ft.

Mr. Ernest Lidgey said that within the last few days he had seen the returns of some cheap milling, which, however, could hardly be compared with the results under consideration, because the ore was mined at less depth, was free milling, contained less than 1 per cent. of concentrates and no cyaniding was done. The total cost including cartage to battery and cost of crushing amounted to 6*s.* 9*d.* a ton, which was perhaps another instance of fairly cheap mining in the colonies.

Mr. S. Vivian said that it would have been interesting if Mr. Thomas had given some information as to how the details were arrived at. At the last meeting Mr. Lidgey had told them that in Victoria they were able to mill and make a profit on ore containing only 2 dwt. of gold per ton. It was a little difficult to see how it could be done in view of the high wages paid in comparison with Dolcoath. He gathered from Mr. Thomas that the miners were not paid by contract but were on day work, and it was difficult to understand how it could be done so cheaply under those conditions.

Mr. Lidgey said that the wages in Victoria were usually 8*s.* to 8*s.* 4*d.* per day. A great deal of work on the low grade ores in question was done by open-cuts, and the stuff carted in at a cost of about 1*s.* per ton. The ore was very friable, and a large amount was put through. He hoped to contribute further details on the subject.

Mr. Alfred James remarked that it struck him, on looking at the analysis, that the labour in connection with the cyanide costs and the cyanide consumed, amounted to a larger sum than was usual for a similar class of ore. It seemed to him that the low general costs were due to the excellent supervision of the little details which in the aggregate went to make up heavy costs. Such items had been kept down to a very minute figure, and the total, in spite of having to pay so much for labour and cyanide, had been kept down to a very small point.

Mr. Franklin White asked how it was that the pumping charges were so low; the depth of the shaft was stated to be 350 ft., and it was unusual to be able to go down so far without having water to contend with. He observed that the cyanide residues were used for filling the stopes; this had been tried in some cases, but the tailings were found to run down into the drives.

He would like to know the time the tailings were under treatment in cyanide.

The paper certainly indicated that the work was being carried on in a very economical manner, and showed that the colonies knew what they were doing. Such papers were of advantage, as they put other men on their mettle to try to show that what could be done in one part could be accomplished equally well, if not a little better, in another.

Mr. Thomas said that Mr. Collins had omitted to state that the cost of labour in Cornwall only amounted to about 2*s.* 6*d.* a day, [Mr. Collins: 3*s.* 6*d.*] whereas in New South Wales it was 7*s.* 6*d.* a day at the lowest. With a similar output to that at the Dolcoath he estimated that the costs would be reduced to 10*s.* a ton. Another question to be considered was that the payment of a manager of a mine such a long distance from home was quite a serious matter, while in Cornwall about 8 guineas a month was the usual pay.

Mr. Collins observed that an underground agent would get 8 guineas a month; the manager of Dolcoath had much more than 8 guineas.

Mr. Thomas, continuing, said that there could be no doubt that the figures given in his paper as to costs were correct, and they represented the actual money spent in the different months. As to the cyanide costs, he thought they were pretty fair, but it was hoped to reduce them during the coming year. The labour costs were certainly rather high.

There was practically no water in the mine, and what little there was they were able to hoist out of the shaft by means of buckets.

With regard to filling the stopes with tailings, they put in two stulls, or scaffolds, at the back of every level and afterwards put from 9 to 10 ft. of rock on the scaffolds so as to prevent the sand from running down.

He was not able to state exactly how long the tailings were in contact, but believed they were in contact with a strong solution from 12 to 14 hours.

They had had more than the usual amount of trouble with regard to labour, and since the formation of the Company, 2½ years ago, no less than 2,000 names had passed through the Company's books.

Mr. Cordner James suggested that Mr. Thomas should give some further details as to the number of men employed in stoping, and the number of trammers and surface men, as it was only by keeping men fully employed that low costs could be attained. He considered that Mr. Thomas's paper was a very interesting one, and showed what could be done under able supervision.

The Chairman announced that the discussion of Mr. Franklin White's paper would be continued. There was not the slightest doubt that dry crushing and cyaniding direct was becoming a very important feature in the treatment of certain gold ores, more particularly in connection with large masses of low grade ore which could not otherwise be treated at a profit.

"Notes on Dry Crushing on the Witwatersrand." By Franklin White, M.Inst. M.M.

DISCUSSION (*continued*).

Mr. Matthew Taylor Brown said that the paper interested him greatly, owing to the fact that the Lisbon-Berlyn Company had been treating their ore in a very similar manner. At a recent meeting he had roughly sketched the treatment to which the ore was subjected by that Company, and he did not propose to repeat what he then said, except to point out that their arrangements were of a much more simple character than that described by Mr. White. They had no conveyors for returning the roughs to be re-crushed, and the grade of the ore was consequently much coarser than that obtained by Mr. White. One of the things which originally encouraged them to adopt dry crushing and direct treatment was an experiment which showed that their ore, crushed as coarsely as 1 in. cube, gave on treatment by direct cyaniding an extraction of about 7 dwt. out of a 9 dwt. ore, which he considered was a very fair result for such coarse crushing. This, together with the facts that they had no coarse gold and that their ore slimed badly, led them to introduce dry crushing tentatively until they finally adopted it entirely. At the present moment their extraction had not come up to that obtained by Mr. White, their last return being about 72 per cent. actual recovery, but they hoped to improve upon that result. Their residues were high compared

with the average on the Rand, but were just about the same value as the residues of companies using wet stamping in the district. Owing to their difficulties in connection with the mechanical part of the process with regard to the wear and tear of the crushers, they had not been able to achieve the perfection they desired in the metallurgical department, but the results obtained had been as good as those they had previously got by combined wet milling, amalgamation, and cyaniding, but these mechanical difficulties had now been surmounted.

One of the most important points in the paper was the proof that dry crushing and direct treatment could be done with a very economical expenditure of horse-power, and the figures given by Mr. White showed that about double the quantity of ore could be treated for a given horse-power to that by wet stamping. At the Lisbon-Berlyn, when their product was coarser than at the Luipaards Vlei, they could make even a better showing than Mr. White, and could crush about 2,200 tons a month, with a 10-hour day shift, working 5½ days a week with 32 H.P. With that horse-power working full time they could not have treated nearly so much ore by wet stamping, especially if a slimes plant were part of the installation. He thought that indicated that the process was economical from the point of view of horse-power, which was an important matter when horse-power was limited. In the district in question they had to depend entirely upon water-power, and in the dry season it was very much diminished. As they were only working a 10-hour day, they had not reached the maximum output for the horse-power available in the dry season, but they had surpassed the output possible by wet stamping. Owing to their process being to a certain extent in a state of transition, they had not yet decided upon the most suitable grade of ore, and he did not care to go into the figures with regard thereto. He wished, however, to draw attention to the point referred to by Mr. Thomas, in his paper read that evening, with regard to the residues being very much the same value with different filling values of the ore. Such had also been their experience, and it had been strikingly illustrated in one monthly assay report recently, in which their residue values had been the same from tanks having a filling assay varying as widely as 10 and 15 dwt. Such a fact showed that the mere quoting of percentage of extraction was often misleading, because, in the case in question, one tank had a filling value of 50 per cent. higher than the average, and yet gave as low residues. In all cases there is an economic irreducible minimum. He thought it was well in making comparisons of the efficiency of processes to make them on ores

having the same assay value to start with and of nearly the same composition as possible.

They always based their calculations on filling assays, and he agreed with Mr. White that this was the only valid basis to go upon.

Mr. R. G. Elwes said that it appeared to him that one of the difficulties in connection with all crushing machinery was the wear and tear of the apparatus, which largely depended on the extent to which the ore could be made to crush itself. Attempts had been made to arrange an apparatus in which the ore should entirely crush itself by one piece crushing the other without coming into contact with iron. In stamp mills there was more wear and tear if a sufficient cushion of ore was not kept between the stamp and the die. He had found in crushing for direct cyaniding that the wear and tear on the jaws was enormous when the ore was crushed too fine in one operation. Since then rolls had been added, and the fine crusher was set to a coarser opening; although the same quantity of ore passed through, the wear and tear was reduced to one-tenth of what it previously had been. If rollers were used to reduce the ore to the grade required by Mr. Franklin White, and set $\frac{1}{8}$ in. apart, it would be found that there would be an enormous wear; if they were set a little wider there would be less wear, although a certain amount of "roughs" would be made, while if they were set still wider the work would become inefficient. He would be glad if Mr. White could tell him what had been found to be the best setting of the rolls, as there must be some point at which the setting was most economical.

Mr. Picard agreed with the author that dry crushing was of considerable advantage in cyaniding the class of ore described. The author gave two reasons for his preference (1) that the ore was delivered when dry crushed into the vat in a thoroughly mixed condition, and (2) that in dry crushing plenty of air was entangled in the ore. Both points were of extreme importance. He had personally experienced a good deal of difficulty in delivering pyritic ores when wet crushed into vats, and had had to exercise great care in the filling. He thought that the reason was that pyrites, when delivered with a large amount of water, showed a tendency to arrange themselves into bunches and layers, which were practically impervious to cyanide liquors. This was probably due to the cubical nature of the pyrites and the way they fractured, so that when delivered with water they formed themselves into almost solid walls. Quartz, on the other hand, crushed very unevenly, and arranged itself awkwardly and angularly in the vats, so that there was a free passage for the cyanide liquor to the quartz. Dry crushing

entirely overcame any difficulty of that kind. With regard to the author's second reason—aëration—it was a matter of great importance in dealing with the cyanide process. He observed that Mr. White did not give his unqualified approval to the method employed in the extraction process, although his disapproval was limited to the intermediate vats. He cordially agreed with him in that respect. It seemed to him that the one point emphasized throughout the paper was aëration, and everything had to give way to it. He agreed with the author that dry crushing was highly advantageous in cyaniding, but the process as set forth in the paper did not seem as satisfactory as it might be. The enormous time taken to treat the ore was also a matter of considerable importance.

After the first treatment of 66 to 70 hours, the ore had to be removed to another tank, and be submitted to a further eleven days' treatment in order to obtain slightly over 1 dwt. of gold per ton. The next point was the large plant required to treat 100 tons of ore per day—exactly 12 tanks, each capable of holding about 100 tons of ore. The great length of time and the large plant required are necessitated by the difficulty of getting air into the ore charge. He concluded that the author in selecting dry crushing was largely influenced by the requirements of the cyanide process. He had decided to use this process for his ore, and he adapted his crushing to suit it. Such careful consideration of the mechanical preparation of the ore was almost as rare as it is desirable.

To summarize the whole position: dry crushing was advisable when the cyanide process was to be adopted, mainly on account of the aëration difficulty; but this dry crushing remedy would not be available for ores of a complex character, which oxidise rapidly. In such cases as the latter, wet crushing was absolutely necessary in order to clean the ore, and some other remedy must be applied to meet the aëration trouble. Supposing for a moment that an alternative remedy did exist, would the author still prefer dry crushing to wet? He endorsed the author's statement, that in many cases amalgamation should not be resorted to at all.

It is almost universally accepted that where there is a gold mine, there shall be an amalgamation battery also, no matter how unsuited the ore may be to amalgamation.

Mr. S. Vivian asked if Mr. White would give them some comparisons of the grading of the products of the ore in question as compared with the products of a stamp battery passing through a similar mesh. He had always heard it contended that the percentage of slimes was very much less with dry crushing, and he took it that it was due to the peculiar construction of the stamp battery

mortar box, which did not deliver the pulp as freely as it was crushed. It was a question whether the manufacturers could not make a mortar box which would give a better delivery.

He thought that the results given of coarse crushing were very interesting. He was acquainted with a mine in Brazil, having an ore very easily disintegrated where by cyaniding, an extraction of from 68 to 75 per cent. was obtained by merely passing the ore through a stone breaker set to $\frac{3}{4}$ inch. It was found, however, that with more finely crushed ore the improved extraction more than paid for the increased cost of crushing. They therefore now crushed to $\frac{1}{16}$-inch mesh.

Mr. Cordner James enquired whether Mr. White had used the Niagara pulveriser for products less than $\frac{1}{2}$ inch, as if so, he would like to hear what his experience had been. He had found it to be unsatisfactory for fine crushing, although he could understand it being capable of doing effective duty in crushing to $\frac{1}{2}$ inch.

He thought that the paper dealt with the subject of dry crushing more particularly in relation to the treatment of the ore, and it would have been interesting to have heard it discussed in the light of some other method of treatment and some other method of crushing. He was sorry the President was not present that evening, as he thought that, as an advocate of wet crushing, Mr. McDermott would have contributed valuable information in connection with the paper, and they would have had to decide whether greater consideration should not be given to the wet process in treating an ore of that particular nature.

Mr. D. A. Louis pointed out that dry crushing was an older practice than wet crushing, but that with the advent of the latter, 200 years or so ago, the former, owing to its acknowledged inferiority, ceased to be practised, except where sufficient water could not be obtained. The position seems to be very much the same now-a-days, for nobody would establish a dry crushing mill, where efficiency and economy were considerations, unless water was not forthcoming or its use overpoweringly detrimental to subsequent operations. His experience of dry milling, certainly some years ago, was that the principal product seemed to be dust, and it was therefore satisfactory to learn that the author had succeeded in erecting a dustless dry mill.

Then as regards rock breakers, in his opinion there was more to be said in favour of jaw crushers than for those of the gyrating type.

The Chairman considered that Mr. White's paper was an extremely valuable one, and was surprised at the amount of information

the author had compressed within the limits of such a short communication.

With regard to the 22,000 tons passed through, he considered that was a conclusive test of the process employed. It would, of course, have been preferable to have had a record of 100,000 tons, but on such a test as the author had conducted one ought to be able to find out everything worth noting.

Mr. White had stated that his plant was not working at its full capacity, and that it was capable of doing more; he would be glad to hear from Mr. White what he would have expected the plant to do supposing it had done full work.

The author, in describing the oxidised ore, did not say whether it contained pyrites or not, and he thought an analysis would give interesting information.

The question of coarse crushing depended largely upon the condition of the gold in the ore. Mr. White distinctly stated that the gold did not occur in the rounded stones, and, therefore, he had every help in getting the gold out, if in a fine state, by coarse crushing. Coarse crushing was coming more and more into use all over the world, and he knew of two companies where coarse crushing on an extensive scale was being practised. In one instance they were crushing to 1 in. and 1½ in., and getting an excellent extraction from an ore more or less of a base nature. The ore in question contained antimony and other minerals, which interfered with ordinary amalgamation, and they could not get the gold out by any other means than by coarse crushing and cyanidation at a profit.

He was glad to note that Mr. White drew attention to the fact that he was able to get his bullion up to a total of 986 in 1,000, and derived great benefit from doing so. It might be remembered that he (the speaker) went into the question some three years ago, and showed the loss companies were making by sending low class cyanide bullion to this country. The refining charges being 4*d.* per oz. of bullion.

With regard to the tables, he thought they contained a great deal of very valuable information.

Mr. W. R. Thomas remarked that with regard to the refining of cyanide bullion, the Myalls Mine were paying the Bank of New South Wales 4*d.* per oz. of fine gold, as it would not pay to refine the gold on the mine. The value of the bullion was about £2 10*s.* to £2 12*s.* per oz.

The Chairman observed that a number of the Transvaal mines were bringing their bullion up as fine from the cyanide process as the "mill gold."

Mr. Franklin White, in reply, said he was very pleased that his paper had elicited such a good discussion, although the task of dealing with the points raised was rather a difficult one.

He agreed with Mr. Charleton's remarks at the last meeting that in making comparisons the surrounding circumstances had to be taken into consideration. It was, moreover, very difficult to give information in a form which enabled fair comparisons to be made, in fact, it was almost impossible to do so. Mr. Charleton quoted figures from the Robinson Mine as compared with those given in his (the speaker's) paper. The question of surrounding circumstances had to be considered when making such comparisons. In his paper he gave the result of what might be fairly called the first considerable trial of dry crushing on the Rand, whilst Mr. Charleton gave the result of work done with a practically perfect mill. He might look back upon the early history of the Rand, and find times when milling cost from 9*s*. to 15*s*. a ton. He thought that his figures would compare very well with those, but that was not the ground he wished to take up. He considered that dry crushing even in its infancy, came very close to the results obtained by the best wet crushing on the Rand. As comparisons had been made between what was effected in wet and dry mills, he wished to draw attention to the figures given. Mr. Charleton quoted the efficiency of the Robinson mill with 950 lb. stamps as 4·6 tons per day per stamp; since that time the weight of the Robinson stamps had been increased, but the efficiency had lately been 4·4 tons per stamp per day. He would not like to say that wet crushing had gone back because of those figures, but he did think he could assert that the returns were now compiled with more accuracy, and that the figures given in the early days were not to be relied upon. It was a difficult matter to question the accuracy of figures, but any alteration in the basis of statistics necessarily affected the figures as a whole, and there was no doubt that on the Rand a good deal of the statistics had been compiled from inaccurate data. The Robinson Company had last year reduced the estimate of tons milled by 5 per cent., because it was found they had previously over estimated. Other companies had done the same, and he thought that when decimal points were being worked out in estimates they should see that the tonnage was right to commence with. It appeared to him that the main feature of metallurgical treatment was to crush the mineral down to such a size as would give the best extraction in comparison with the cost of wear and of horse-power, and he intended to confine his reply more or less to those matters, and leave the question of labour, as that entirely depended upon the magnitude on which the work was

conducted, the care bestowed on the arrangement of the plans, and the local conditions. Whether the ore was crushed wet or dry the cost of handling should be the same.

Mr. Charleton quoted a figure showing that the cost of horse-power at the Robinson was less than the cost of power at Luipaards Vlei; that was another instance of the difficulty of making comparisons, as he (the author) did not include electric light and electric power in his figures, but they could, of course, be added. His figures included fuel, engine-drivers, supplies connected with the engines, together with proportion of general management and secretarial charges. The Robinson cost was for fuel alone, and, therefore, the comparison of 8·24*d.* as against 8·7*d.* was hardly a fair one. He observed that in a subsequent year the cost of fuel at the Robinson had run up to over 9*d.* per ton milled. At the Robinson they had to pay about 10*d.* for concentration and mill water charges, whilst in dry crushing they did without these. The rock breaker cost at the Robinson was lower than that at the Luipaards Vlei, but this was a matter which chiefly depended on the magnitude of operations. In the costs at the latter mine, they included the whole of the charges in connection with native labour, while in the Robinson accounts such charges were put down to "general charges"; another instance of the difficulty of making a comparison between the accounts furnished by one mine and those of another.

Messrs. Hatch and Chalmers had been quoted as an authority adverse to the employing of dry crushing, but he did not see anything in their book which gave the idea that this opinion was based on their own experience. They quoted several well-known men, amongst others, Mr. Butlers, Mr. Bettel, and Mr. Crosse. Mr. Butlers considered that dry crushing in order to compete with wet should be able to crush ore to 600-mesh at a cost of 2*s.* a ton. It was difficult to understand why 600-mesh should be chosen, as dry crushing appeared to show that equally good extraction could be got with 200-mesh.

The milling cost on the Rand, including water and concentration, at present varied from 2*s.* 11*d.* to 3*s.* 3*d.* per ton in mills with from 120 to 160 heads of stamps, and the small mill at Luipaards Vlei could naturally not compete with that figure. The main points were horse-power, wear and tear, and extraction. Mr. Bettel, quoted by Messrs. Hatch and Chalmers, stated that the wear and tear in dry crushing mills was so great that magnets would have to be used in order to separate out the finely divided iron from the mass of crushed ore, as chemists considered that the finely divided

iron was very detrimental to the action of cyanide. The wear and tear for shoes and dies at the Robinson mill was close on 0·75 lb. of metal per ton of ore crushed, while the wear and tear at Luipaards Vlei was a little over 0·8 lb. He thought that these figures showed that there was not much in favour of Mr. Bettel's statement, especially when a reference to the tables given in his paper indicated that the greater part of the wear and tear was due to really unworn parts in the Niagara mill, and not to the rolls. He considered that the inference was clear that if rolls had been used throughout instead of the Niagara mill, the wear and tear would have been less than 0·75, which was now considered good work in wet stamp mills. Another of Mr. Bettel's difficulties was the cost of drying the ore, which he said would be in proportion to the moisture present. That statement was correct; but as there was not sufficient moisture present to make drying necessary, no cost was incurred. Mr. Brown, in his interesting statement in connection with the Lisbon Berlyn, did not mention whether anything was spent on drying the ore, and he (the speaker) presumed there was no expenditure. Another point supposed to condemn dry crushing was the expenditure in cyanide, as 1 lb. was said to be required against 0·46 lb. in the Robinson. He had been able to bring the consumption of cyanide down to 0·33 lb. per ton, and on one occasion it was reduced to 0·24, and he certainly thought they were entitled to credit for that result.

Mr. Crosse referred to a test which, "provided it could be done cheaply, would no doubt be a good process." The principal idea in this case was to replace water as a vehicle for carrying away the crushed ore by air introduced into the mortar box from behind, and to blow out the crushed ore as fast as it was made. This test was really made by the speaker, but the result was that the crushed ore instead of being blown out where the inventor wanted it to go, and that was through the discharge provided for that purpose, it was blown out of every little crevice, and the result was something like that mentioned by Mr. Louis—there was dust everywhere. He thought that any engineer would understand that trying to blow air into a mortar box to make the ore go where it was wanted to go, was rather a difficult proposition, and therefore Mr. Crosse's objection could have no weight at all as against a well arranged plant. It was impossible to finish the experiment as they were unable to work in the mill on account of the dust and the irregular size of the crushed material.

With regard to extraction, Messrs. Hatch and Chalmers considered that 93 per cent. of the gold contents should be obtained by wet

milling, amalgamation, cyaniding of tailings and slimes. They had always been taught on the Rand that an extraction of 93 per cent. was easily obtained, and there were plenty of statistics made, the purpose of which was to show that such was the case. But in the report of the Jumpers Deep, recently published, it was stated that the extraction was 80 per cent., and the chairman of the company considered it to be a good result. The mill in question was a modern one, with a good cyanide and slime plant. It was run by the best men on the Rand, and under the supervision of the best mining house—the Rand Mines. The reason for this apparent decrease in extraction was that the Jumpers Deep had commenced to ascertain the value of the ore going into the mill by assay, and to take correct account of the weight; as a result the extraction had come down from 90 to 80 per cent. He thought it reasonable to claim that the Jumpers Deep report should be taken as a standard, as it would not have been published if those in charge had not been satisfied that it was the result of straightforward work. In the report the theoretical extraction from amalgamation was given as 52 per cent., while that actually obtained was 51 per cent. From the sands and concentrates 33 per cent. of the assay value should have been obtained, but only 28 per cent. was extracted. From the slimes 0·4 per cent. was obtained as against 0·74 per cent. Instead of extracting a total of 86 per cent., they obtained 80 per cent., although no doubt the final clean up would give them 1 or 2 per cent. more. He thought the figures he had given showed that dry crushing, which claimed, from results obtained in what was really only a trial stage, to extract from 80 to 85 per cent. was pretty well on a par with the wet mills, if not ahead. Again, in the report of the Crown Reef Company for 1897 it was stated, with regard to the slimes, that they should have obtained 77 per cent. of the assay value, but they only obtained 53 per cent. He thought their extraction was fairly good, but they had not really put the number of tons of slimes they had calculated into the tanks. When the tonnage put into the mill bins was "estimated" there was no check upon it. The cyanide man had a check upon what went into his vats, and now that the slimes man had a check there was a hope of being able to arrive at the actual amount of ore put into the mill. In his opinion the slimes man at the Crown Reefs was debited with about 12,000 tons more than he actually received, hence the low extraction; if they altered his tonnage, however, they would have to alter the mill tonnage, and that altered the cost per ton. But if the tonnage were really correct, some 2,000 ounces were unaccounted for.

A reference had been made to a remark of Mr. Thomas that the extraction rose in proportion to the value of the ore. That had been his experience. He really could not say why low grade ore seemed to give nearly the same residue value as ore of higher grade.

Mr. Elwes had asked what was considered to be the economical point of adjustment of the rolls. He could only say that it was a mistake to press them too tight and try to crush the ore at one operation. Gradual reduction seemed to be the best—work down from one machine to another in stages.

Mr. Picard seemed to think that he (the speaker) was not entirely in favour of the cyanide process. As a matter of fact, he considered that Mr. Feldtmann's process was the right one, and the main difference between it and what was usually carried on on the Rand was that Mr. Feldtmann advised that after the cyanide solution had been run on to the vats it should be allowed to remain in contact for a short time and then run off and the vats allowed to remain damp. The ordinary practice was to leave the vat full of cyanide solution, the effect of which was to stop the entrance of more air into the mass. Feldtmann's process simply facilitated the entrance of air.

He was unable to answer Mr. Vivian's question as to the grading of the products of wet and dry crushing through a similar mesh. He was unacquainted with any instance on the Rand of ore being milled to 200-mesh, but 600-mesh gave about 50 to 60 per cent. of a minus 90 product, while his product would run about 20 to 25 per cent.

With regard to Mr. Cordner James's remarks, he had given the amounts of material used in the Niagara machine, and the Gates rolls, and must leave them to draw their own conclusions. There was no doubt, however, that the rolls worked admirably.

He was sorry to hear that the old question of dust had been raised. They fitted up the mill with exhaust fans, settling chambers, &c., and after three days' work he went to get a large sample of the dust in order to examine it. On asking the mill man to get him some out of the settling chambers, he was told that they had stopped using the fans as there was no dust to draw away.

He regretted he was not able to give the results of more than 22,000 tons, as he left the property at the completion of that quantity. He thought, however, that the amount treated gave a fair idea of what could be done.

Comparisons had been made between the Luipaards Vlei and other mines, and he thought it would be interesting to give some figures with regard to another dry crushing mill in the Transvaal—

the Afrikander. At that mine they crushed a quantity of hard pyritic rock, and the total extraction at the completion of the test was 72 per cent., while 3 per cent., due probably to errors in weights or assays, was unaccounted for. The reason for the low extraction was the wear of the crushing machines. On the completion of the test 27 per cent. of the rock would not pass through ¼-in. mesh. At the commencement, when the crushing surfaces were in good order, the first six charges gave an average theoretical extraction of nearly 86 per cent. He had found no difficulty in keeping the crushing surface of his rolls in good order, and, therefore, the extraction obtained by the Afrikander people should have been maintained throughout, if their machinery had run properly.

The Chairman had inquired whether pyrites was present in the free milling ore. There was a small percentage of pyritic ore from development work, all of which went to the mill.

A remark had been made to the effect that as the gold was not in the pebbles it was somewhat in their favour, seeing that it was not necessary to grind that portion of the rock so fine; on the other hand, the great portion of the gold on the Rand was in the pyrites, which he considered was rather worse. He would sooner get gold out of quartz than out of pyrites; as a matter of fact, the presence of the gold in the pyrites was the reason of the long treatment they had to give it.

"Mica Mining in Bengal, India."

By A Mervyn Smith, M.Inst.M.M.

Position.—The mines described are situated between 85° to 86° 30′ east longitude, and 24° 25′ to 25° north latitude. The country is made up of a series of parallel ranges of low hills some 400 ft. above the surrounding country and 1,200 ft. above sea level. The direction of the ranges is east and west nearly, they form the boundary between the Hazaribagh district on the south, and the Gaya and Monghyr districts on the north being part of the Bengal Presidency, India.

Geology.—The country is made up of gneiss, grading into mica schists, tourmaline schists, hornblende rocks, quartzites, and a coarse felspathic rock resembling conglomerate with intrusive dykes of fine grained diorite.

The hornblende rocks closely resemble the diorite, and weathers into similar rounded boulders. But this variety passes into a much coarser kind in which "blebs" of white felspar are so close-placed as to give it a "banquet" appearance. The mica schists are strongly developed in this area and are made up chiefly of white mica (muscovite) with here and there a little black mica (viotite) and red mica (lepidolite).

This rock is highly schistose, breaking up into thin laminæ and consists chiefly of small plates of mica, cemented together with a slight admixture of felspar and quartz. It is this rock that contains the lodes that yield the mica of commerce.

The gneiss and other foliated rocks have a nearly east and west strike (about 5° north of east and south of west), and underlie to the north at an angle of about 75°. These rocks are classed in Indian geology as among the younger gneisses.

Pegmatite Veins.—Running with the bedding of the schists, from east to west, are numerous veins of granite (pegmatite) varying in width from a mere thread, to 20 ft. The schists are in places faulted or thrown out of their normal strike. The pegmatite veins are subjected to the same "faulting" and frequently at the point of

"faulting" short north-to-south veins or "cross-courses" of great width have been formed.

The veins are made up of amorphous masses of quartz, large crystals of pink felspar (orthoclase) and crystals or "books" of mica (muscovite). The quality of the vein varies with the nature of the adjacent rock. Near to the quartzites, the vein stuff is nearly pure quartz with little felspar and mica. Where the adjacent rock is highly felspathic, pink felspar crystals are the chief constituents of the lode; while we may look for a large predominance of mica, where the including rock is mica schist. This fact is of importance, as a knowledge of this peculiarity of segregation in the vein-matter enables the miners to descriminate between spots that will yield more or less mica.

In addition to these three chief constituents of the pegmatite veins, the following accessory minerals are sometimes found—tourmaline, garnet tin-stone and a little columbite. The tourmaline is at times strongly developed. Near to the tourmaline schists, large massive crystals of black tourmaline are not infrequent in the lode, or occasionally crystals of tourmaline penetrate the mica books across the planes of lamination and destroy the value of the sheets.

The pegmatite veins in this district are apparently of aqueous origin and due to segregation. The following are some of the data pointing to this conclusion:—(1) In many places the veins die out in depth and in length. (2) The vein-matter partakes of the quality of the including rock, quartz, felspar, and mica predominating as the lode traverses rocks containing these minerals in excess. (3) The crystallization of the vein-matter varying within very short distances not being at all homogeneous, as one would expect in an igneous dyke. In places the crystals of mica are 2 ft. long, 18 in. wide, and 9 in. thick. Within a few hundred feet on either side of this spot the mica books are small and worthless.

Age of the Mines.—This zone of mica-bearing rocks has been worked for centuries by the Hindus. Mica is largely used by them for ornamental purposes such an inlaid work, tassels, flowers, toys, banners. The large clear plates are much used by native artists for portrait painting, as being extremely durable and not affected by heat or damp and impervious to the attacks of insects. The waste mica is ground to a coarse powder, and after being mixed with starch is applied to cheap cotton cloths to give them a sheen.

The great marts for mica for native consumption are Patna and Delhi. Dr. P. Breton who visited these mines in 1826, found as many as 5,000 people at work at different mines. In 1849 Dr. McClelland gives the out-put as 100,000 maunds (a maund

being 82 lbs.). In 1863 the statistical account of Bengal says 10,000 maunds were exported.

The Native Methods of Mining.—The native mines are of the most primitive kind. Open cuts along the out-crops of the pegmatite veins where books of mica are seen in any quantity. These cuts are continued down 20 or 30 ft. till the sides become dangerous. No timbering is used to keep the sides from falling in, and frequently accidents happen and the miners are buried beneath the fallen walls of the reef. Where exceptionally rich stuff is met, and the vein-stuff is decomposed and soft to some 50 or 100 ft., inclines are put in, and follow down the rich shoots of mica in a most tortuous course, zig-zaging about from side to side with the "leads" of rich ground. Long lines of women near to each other and placed in double row, from the water level to surface, hand out earthen pitchers—called "gurrahs"—to one another; the full pitchers are handed up one line and the empties down the other. As many as seventy women are sometimes placed to remove the water from a mine not 35 ft. in perpendicular depth. The incline, of course, was much more than this in length.

Where the reef is highly felspathic and of large size, decomposition extends to quite a hundred feet or more, and it is here that their largest mining operations are carried on. In order to ventilate the inclines and draw out the mica perpendicular shafts about 2 ft. in diameter are put in. The author counted as many as thirty of these circular shafts along the strike of the reef and within a few feet of each other.

The miners are a local tribe called "Bandathis"; the men, women, and children all working at the mine when they have no agricultural work in the fields. Work is only conducted in the dry months (November to May). Immediately the rains set in, they return to the tillage of their fields. Work is only conducted in the day, beginning about 8 A.M., and giving over at dusk. Where the veins are hard yet sufficiently rich to pay for the labour, large fires are kindled against the face of the lode, and when the vein stuff is sufficiently heated, water is thrown on to it, when the sudden cooling causes the rock to shrink and crack. Into these cracks wedges of soft iron, locally manufactured from the magnetite iron ores common in this neighbourhood, are driven, and large boulders detached. The exposed books of mica are chiselled out and taken to surface. The books of mica are split into sheets of about one-eighth of an inch in thickness. All the rough edges and flaws are trimmed off by means of a sharp sickle (called "hasawah"), and then sorted according to colour and size. The sheets are in some cases

very large, 24 in. by 18 in., being the largest in this district. In the Itakuri Mine, Nellore district, Machas, plates 60 in. by 40 in. have been obtained.

Present Method of Work.—It is only within very recent years that Europeans have taken up the mica industry. Mining is still conducted on purely native methods described above; and although 250 miners are at work in this district, on not a single one is machinery of any kind used, and certainly on none of the properties visited by me this year (1898) was there the least attempt at European methods of mining. The same wasteful, slow, and laborious system practised by the natives for hundreds of years is still in vogue. Men are sent out during the rains to search for likely looking outcrops of mica. The mica-schist being softer than the other members of this series of rocks, is eroded into valleys—the more quartzose beds making the ridges of hills. A certain amount of alluvium and talus from the adjacent hills covers the valleys.

After exceptionally heavy rain, this surface deposit is washed off, and with it the decomposed felspar of the pegmatite veins, leaving exposed tufts of partially decomposed mica. The natives call these tufts "foo-foo," and believe that they grow during the rains, as they cannot account for their exposure at surface after a heavy shower, where there was nothing apparent before the rain. Having discovered several of these "foo-foo" spots, these are marked off for mining operations as soon as the dry season begins. Parties of men, women, and children are set to work on these outcrops, and the books of mica dug out, packed in loads of about 30 lb., and brought in at dusk to the central store.

Here several skilled men are seated on the floor trimming the mica before it is packed away for market. Before each man is a stout peg, driven firmly into the ground and protruding about 18 in. The books of mica are first split into plates about one-eighth of an inch in thickness, the mica easily splitting into laminæ of any thickness. The trimmers are provided with sharp sickles, and the point of this knife is used for opening the sheets. Imperfect laminæ are now pealed off the plates till both surfaces show a clean, even face. The plate is supported against the side of the peg, and the sickle drawn downwards to trim off the jagged ends and irregularities along the edge of the sheets of mica.

After trimming, the plates are sorted for the European market, the United Kingdom and America being the chief buyers. The sheets are first sorted according to quality, four kinds being recognised by the dealers.

1. Ruby mica, hard and tough.
2. White transparent mica.
3. Discoloured and smoked.
4. Black mica and flawed.

If 8 represents the value of ruby mica, 4 would be paid for white, 2 for discoloured, and 1 for black and flawed sheets of equal size.

The sizing is as follows :—

Specials, sheets measuring more than 50 sq. in.

No. 1.	Sheets of from	36	to	50	sq. in.
„ 2.	„	24	„	36	„
„ 3.	„	16	„	24	„
„ 4.	„	10	„	16	„
„ 5.	„	6	„	10	„
„ 6.	„	4	„	6	„

Quoting from a recent sale sheet at an auction in London, the following prices were realised :—

	s.	d.	
Best ruby, No. 1	6	8	per lb.
„ 2	4	0	„
„ 3	2	0	„
„ 4	1	0	„
„ 5	0	4	„
„ 6	0	2	„

Specials bring as much as £1 a lb. according to size of sheets.

The sheets are trimmed irregularly into any shape they will take to clear them of flaws. Should square, rectangular, or diamond-shape sheets be wanted, a special rate has to be paid for these to allow for the great waste.

The sheets are packed in boxes of 1 cwt.. and transported on carts to the nearest railway station, 100 miles distant. From thence it goes to Calcutta and shipped to London or the United States of America.

Market and Out-put.—During the year 1895–96, 8,913 cwt. of mica were exported from India, of which the mines under review contributed 8,835 cwt., valued at Rs. 995,000 or £66,000. The yield of the mines may be placed at double this figure, one-half being required for the Indian market, and these chiefly of the inferior kinds.

The export trade is a growing one, as the following figures will show :—

Exports.

	1892.	1893.	1894.	1895.	1896.
Bengal ...cwt.	2,298	3,310	4,843	5,126	8,835

Hard, sound, ruby mica is much used for the doors of glass and steel furnaces where extreme heat is required. The mica is not affected by the extreme heat, and thus enables the workmen to look into the furnace and watch the crucibles of molten matter. Thin sheets are made into chimneys for incandescent gas burners; also for fire screens and for electrical purposes, as mica is an extremely bad conductor of electricity.

A method of cementing sheets of mica into a cardboard has been recently patented in Germany, and this cardboard is used for coating boilers, mica being a non-conductor of heat. It is also manufactured into helmets for fire-brigade men, packing for fire-proof rooms, &c. Thinner sheets are made into envelopes for wrapping valuable documents.

Quantity of Mica Available.—At present, India yields the bulk of the mica consumed in the arts, and of this quantity nearly all comes from the district described.

The pegmatite veins in this locality are numerous and of large size, and are in places extremely rich in mica, nearly one-third of the vein-stuff being of this material. The quantity of mica available is practically inexhaustible.

The present method of mining is wasteful in the extreme, and fully nine-tenths of the mica extracted from the mines is injured and rendered unfit for export.

Thousands of tons of waste mica are to be seen at the mouths of the mines and at the dressing floors. Probably the new patented method of cementing mica into cardboard may create a market for this.

There cannot be the slightest doubt that when European methods of mining are introduced, the cost of production will be materially reduced, the waste will be much less, and a better quality of mica will be secured. At present only the decomposed, or partially decomposed, vein-stuff is mined. Here the mica must also undergo a partial decomposition, but of course not to anything like the extent of the felspar. In the hard vein-stuff the mica is hard and tough, and this class of mica commands a better price.

Some of the veins yield black mica (biotite). This is largely used as a drug by Hindus and Mohammedans. Reduced to a powder, it is supposed to be very efficacious in cases of dysentery.

The sheets of mica are at times very queerly marked. In places

one-half of each sheet will be muscovite and the other half biotite, the line of division between the two colours being a perfectly straight line, and there being no apparent change beyond the colour in the uniformity of the sheet. Other sheets again are marked with a chequered pattern in black lines, the lines being due to magnetite. Again there are dendritic inclusions of white quartz between the laminæ. All these markings take from the value of the sheets. The most esteemed colours are pure ruby, amber, light green, transparent white. There is also a silver white, which the natives prize for inlaid work.

A demand for Indian mica has lately sprung up in the United States of America. In 1895 the quantity taken was 1,900 cwt., while in 1896 it was 5,076 cwt. Indian ruby is found to answer the purposes of furnace work better than the locally obtained mica.

SIXTH ORDINARY MEETING, 15th March, 1899.

Mr. WALTER McDERMOTT (*President*) in the Chair.

The Minutes of the last meeting were read and confirmed.

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"On Crushing and Concentration at Dolcoath Mine, Cornwall."

By R. ARTHUR THOMAS, M.Inst.M.M.

PREVIOUS to the installation of frue vanners, the usual system of tin dressing in Cornwall was in vogue at Dolcoath Mine. This being so well known it is unnecessary to do other than briefly refer to it, and to note a few changes which have gradually led up to the introduction of efficient automatic machinery.

After stamping, the pulp was, up to a few years ago, passed into "strips" immediately in front of the Cornish stamps, these strips being troughs about 20 ft. in length and 18 in. in width and depth, having an incline of about 1 in 30. A partial concentration was thus effected, and a considerable proportion of the slimes separated by overflowing.

Three products resulted from this operation, which were further dressed in buddles. The slimes passed direct into settling tanks, and after a sufficient time had elapsed for proper settlement, were treated over self-acting frames, the concentrates from which were again settled and further treated by girls over dead frames. The first products from this washing were "buddled" until the heads contained about 25 per cent. black tin and then calcined.

The "strips" have been entirely dispensed with, the pulp now being passed into a classifier, the fines from which are concentrated on a revolving table, thus dispensing with the slime treatment mentioned above. The table has a slow revolving bed sloping towards the centre, and is divided into two parts, in order to make two classes instead of one as previously. The inclination varies but as a rule is 1 : 12.

The overflow from the revolving table on being settled contains

enough slimes to be automatically treated at a profit over self-acting frames.

The coarse sand from the classifier is run through an ordinary buddle from which three products are obtained: (1) heads, (2) middles, and (3) tails.

The "heads" are usually sent at once to the calciner. The second and third products are again "buddled" until a sufficiently high product, about 25 per cent. black tin, is obtained, which is calcined.

Brunton's calciner, the type in use, is a rotary furnace revolved at varying speeds, depending on the nature of the stuff to be roasted.

After calcination the concentrates are again buddled and "kieved," until a marketable product is obtained.

The buddle separation shows fairly good results, but entails an excessive amount of labour; the system can only be defended on the ground of being already established, or the difficulty in obtaining the necessary capital required for making a change for the better. And again, the actual is greater than the apparent loss in treatment, because of the many re-handlings of intermediate products. As a result of the introduction of more automatic machinery, apart from the frue vanner plant, considerable reductions have been made in the cost of extraction.

About 12 months ago experiments were made with the idea of still further reducing this amount of labour, and getting by automatic machinery sharper separation, and consequently more economical results. In these experiments two systems of dressing were tried:—

(1) Classification, and subsequent treatment of the fines over a convex revolving buddle, and the roughs over two percussion side-inclined belts.

(2) A single treatment over vanners without classification.

In order to test the capacity and efficiency of the various machines the trials were carried out on tin-stuff varying in character and value, and extended over a period of three months.

Two 4-ft. vanners took the pulp from a five-stamp mill, the ore in every case being carefully weighed, sampled, and assayed.

Ten stamps supplied the pulp for the other machines. The concentrates from all the machines were weighed, sampled, and assayed. The percentage of extraction was thus accurately ascertained.

With the first system very good results were obtained, but too many products requiring further re-treatment, and further consequent losses, were produced.

As the vanner in taking the pulp direct from the stamps without any handling or classification, produces a good clean concentrate at one operation, which is higher than that made by the other machines, and with small loss in the tailings, it was found to be the most suitable.

The mill crushing for this vanner plant consists of 60 stamps of the ordinary Californian pattern, and fitted with suspended Challenge feeders and a few roller feeders.

Forty of them have a falling weight of 850 lb., the other 20 heads 1,050 lb. each.

In addition to the 60 heads, there are two heads of Husband's pneumatic stamps, having a capacity equal to about 20 heads of the California 850 lb. stamps. The mill is, therefore, equal to 80 stamps.

The motive power is a horizontal compound condensing engine, having cylinders 17 in. and 28 in. and 48 in. stroke, made by Price. The horse power developed is 143. Steam is supplied by three Cornish boilers, each 6 ft. by 30 ft., working at a pressure of 100 lb. per sq. in. These boilers have recently been fitted with Meldrum furnaces, in order to increase their steaming power.

Steady driving being absolutely essential for the vanners, an engine 10 in. by 14 in. stroke, vertical type, was erected to do this work alone. This engine condenses in the compound engine condenser. A pumping engine (horizontal) and an electric light installation are used in connection with this plant. The engines are in one engine room, and are under the control of one man in each shift.

The ore is screened, selected, and crushed to 1½ cub. in. in the stone-breaking and sorting shed, near the main winding shaft, and is then trammed into the mill ore bins. Owing to the extreme hardness and toughness of the ore, great difficulty has been experienced in getting jaws to stand. Manganese steel is now being used with good results.

The stamps are set with an 8-in. drop, and are run from 85 to 90 blows per minute. Punched copper screens (round holes) are used in the mill, and are No. 37 Cornish mesh, equal to 27 wire mesh. As the bulk of the loss of tin occurs in the slimes, it is a point of great importance to arrive at the limit of the mesh of the screens and weight of stamps with efficiency of subsequent extraction. Experiments are now being made to determine whether by still increasing the weight of the stamps more slimes will be produced.

The average daily crushing of the whole mill is 125 tons (2,240 lb.).

Various forged and cast steel shoes have been tried, but hard

cast iron made by local founders have worked out at a lower cost per ton, allowing for re-melting of the old castings when worn down.

The tin ore is exceedingly difficult to crush, as it contains but a small percentage of friable quartz, and a very large proportion of tin capel, which is both hard and tough.

The poorer grade ores are the hardest, and contain the greater proportion of fine tin. With the present price of black tin (£62 per ton), 1 per cent. ores are profitably worked with this plant. The richer ores crush faster, and need more careful concentration.

The vanner plant consists of 27 6-ft. machines with plain belts, having Brownell patent flanged edges.

In the preliminary trials of corrugated belts against plain belts, the former were found to bring up too much coarse material, reducing the percentage of tin in the concentrates.

An ample number of vanners has been laid down so as to efficiently handle the pulp when the mill is crushing the highest grades of ores.

The method of dressing adopted is a single treatment on vanners, with re-treatment of the tailings after classification, the slimes on revolving buddles, the coarse by concentration and pulverising.

It is questionable if the vanners are now sufficiently bedded with pulp to give the best results.

One side of the mill, consisting of six batteries of five stamps each, is supplied with 12 vanners, the other side also 30 heads, has nine vanners, or three to each 10 stamps. The pulp from the two heads of pneumatic stamps is fed on to six vanners.

The richer grade ores, sometimes running as high as 15 per cent. of black tin per ton, are crushed on the side of the mill having the 12 vanners, two to each five stamps. The vanners are run at 192 revolutions per minute, and are spaced at 10 ft. 6 in., with a 6-ft. passage way between their heads.

The main water service (3-in. and 4-in. galvanised piping) is brought over the passage way on the girders carrying the counter shafting, with 1-in. branches to each machine; similar branches are carried down at either end and in the middle of the shed, for washing down the floor, which is sloped $\frac{1}{4}$ in. to the foot from the centre of the passage way.

The vanners are set on sills, on pitch pine 10 in. deep and 8 in. wide. These were bedded in 2 in. to 3 in. of concrete, in order to make them bear all along their length, and to give a wider bearing on the ground, which was "made" under the outer row of machines.

A launder having two compartments, each 10 in. deep, is laid all around the vanner floor. The inner one catches the washings from

the floor, and the outer one receives the overflow wash water from the belts. This launder is set perfectly level, so that the water can be drawn off at any point of the circuit.

The tailings launders are under the floor, and have a grade of $\frac{3}{4}$ in. to the foot. They empty into a cross launder, which takes the tailings to the classifier.

Wooden swing gates, which can be held in any position by a wedge, are put in the launders, to regulate the amount of pulp to each machine, or if necessary to cut it off entirely from any machine during temporary stoppage.

The delivery from the launder to the distributer is by means of a $1\frac{1}{2}$-in. hose, which has brass hose couplings at each end, screwed firmly on to corresponding parts that are attached to the launder and distributer.

A short launder from the battery delivers the pulp on to a "grating" box (punched zinc screen) at the edge of the battery floor, to prevent "gravel" from getting on to the vanner belts.

Two men in each shift (four in all) attend to and regulate the vanners, in addition to which there are two men employed in repairing the machines. These six men per 24 hours attend to the whole of the concentration work, as far as the vanners are concerned. There is a general foreman over the whole mill.

The concentrates are drawn and assayed every day. Samples of the tailings from the belts are taken every day, and after 24 hours are allowed for settling, the water is decanted, and the sample assayed. The average value of the concentrates is 50 per cent. black tin per ton, the black tin containing 65 per cent. metallic tin.

A higher concentrate can be made, but it is found that the loss of fine tin in the tailings increases when making a higher grade concentrate.

About one ton of slime tin is produced per month from the settling boxes taking the overflow wash water from the concentration troughs. This product is excessively fine.

The loss of tin in the tailings amounts on the average to 3 lbs. per ton of stuff.

The percentage of extraction thus shown on the vanners is a little over 94 of the gross contents, as ascertained by assays with the vanning shovel.

The concentrates from the vanners are calcined, and further treated by washing, in order to get a marketable product. Endeavours have been made to produce a concentrate that can be sold direct to the smelters without burning, but as yet unsuccessfully, as the concentrates contain too much fine waste and coarse sand,

which reduces the parcel beyond the payable limit. This, however, may yet be worked out.

The cost of producing a concentrate containing about 25 per cent. black tin for calcining by the old method at this mill was 1*s.* 3*d.* per ton of stuff stamped. The cost of treatment by vanners, including all labour, steam, stores, maintenance (at the rate of 6½ per cent. per annum), interest on the capital expenditure (5 per cent. per annum) is under 6*d.* per ton.

The vanner produces a concentrate that contains twice as much black tin per ton as by the former system, the after treatment of which is in proportion reduced; and the total loss in the whole tailings, allowing for the former retreatment of intermediate products, is doubtless less under the new system than with the old.

DISCUSSION.

The President thought that Mr. Thomas's paper was a very valuable contribution to the *Transactions* as being, in a brief form, the results of experiments made in Cornwall, and affording comparisons between the old, as compared with the modern, systems of concentration, and giving figures and reasons for the decision to alter the system to the present one. The increased percentage of block tin, as shown in the concentrates produced was in itself a strong argument for the method of treatment which had been adopted, and the loss in tailings was certainly less than with the old system.

The paper raised a number of important points in connection with concentration and, amongst others, that of classification. They had been so accustomed to hearing that concentration without classification with modern scientific systems was ridiculous, that there would doubtless be some advocate of a still greater improvement on the present system at Dolcoath by the adoption of classification. The point which struck those who visited the present Dolcoath works, in comparison with the old ones, was the small amount of labour, the extreme regularity of the work, and the absence of all dirt and carelessness about machinery. The new arrangement was particularly noticeable for its neatness of design, and the whole mill was kept in a condition similar to that which would prevail in a good engine room.

He hoped to hear some discussion on the question of the capacity of the mill, as it appeared that it was small for the number of stamps employed as compared with other parts of the world.

Mr. J. H. Collins hoped that the author would at a later period furnish them with more experimental details; the paper gave the conclusions arrived at in a very clear manner, but it did not set forth the data upon which the conclusions in question were arrived at. It was stated that the percentage of extraction shown on the vanners was a little over 94 of the gross contents as ascertained by assays with the vanning shovel, while the cost of the vanning operation was given as being under 6*d.* a ton. It would be interesting to know the whole cost of crushing and dressing by the modern plant, as compared with the old one. As regarded vanning, if the figures could be relied on, the result must be regarded as highly successful, a saving of 94 per cent. of the total tin, at a cost of under 6*d.* a ton, was certainly remarkably good. He would like to ask Mr. Thomas if he was sure that the actual loss of tin in the tailings was only 3 lb. to the ton, or say 4½ per cent. of the tin in the stone. He was told that the new vanning plant was very popular at Camborne, not only with the Dolcoath staff but also with the "streamers" in the river below, which had largely increased their returns since it went to work. If this was the case it did not necessarily condemn the vanners, but it pointed to the necessity of a more thorough treatment of what went over the vanners. He thought it was a very great mistake to suppose that any one type of machine, however good, could do everything, and having got a thoroughly good concentration of the bulk of the tin on the vanners, it seemed to him that the stuff that went over the vanners was still of sufficient importance to require treatment in some other way, and as far as he was aware, there was no better method than using the revolving tables, upon which Mr. Thomas appeared to cast some slight in his paper. The Dolcoath Company possessed a very complete and fine set of revolving tables, and he certainly thought that the stuff going over the vanners should be more completely treated than it had been up to the present. It was also possible that the coarser sands, which consisted partly of clean oxide of tin and partly of quartz, or tourmaline, would be best concentrated by jigging. If he had to deal with such large quantities of stanniferous material he should certainly be disposed to try separation and classification into slimes and sands, concentrate the sands by jigging, and treat the slimes on the revolving tables, which, as far as he was aware, were the best machines for concentrating slimes. The material that went over the revolving tables should be still further treated on the old-fashioned dead frames, which were very cheap in working and exceedingly effective.

Mr. W. R. Thomas said that he thought the author, in referring

to revolving buddles, really meant revolving tables. Personally he had heard nothing said against the tables, in fact, on the contrary, they were spoken of very highly by those in charge of the dressing operations. He thought the fact that the tables were driven by water wheels rendered them less efficient, as the motion was very jerky—they revolved all right for a time, and the water flowed beautifully, but they would suddenly stop and then run down.

He would have liked some information with regard to the duty of the Californian stamps, as compared with the old Cornish stamps; he understood they had about 250 of the latter at work. He found from the reports issued every month to the shareholders that about 6,000 tons a month was crushed by the Dolcoath Company, which worked out to about 250 tons a day; the Californian stamps, of which there were 80 of 850 lb. each (which was very light for a hard rock like tin capel) were doing a duty of 1½ tons, representing 125 tons a day—the balance was evidently being crushed by the 250 or 300 Cornish stamps. He did not see how it was possible for anyone to stand up for the Cornish stamps as compared with Californian ones.

He thought that the paper might have gone much further, and would have liked to have seen itemised costs.

Three kinds of stamps were in use—the Cornish stamp, the Californian stamp, and the Husband stamp. He had had some experience of the latter, and breakages made a heavy item in their maintenance; he had known cases which cost £60 to £100 to repair. It would be interesting to know the duty from each kind of stamp.

With regard to the cost of dressing the tin, the figure of 6*d.* given in the paper did not cover the cost by any means. It would be interesting if Mr. Thomas would give them some information on the point.

Mr. J. H. Collins remarked that he could not understand how Mr. W. R. Thomas could describe the working of water wheels as jerky, seeing that it was known to be the smoothest motion in the world.

Mr. W. R. Thomas observed that at "Cook's Kitchen," at Dolcoath, it was decidedly jerky.

Mr. W. Thomas thought that if classification had been adopted with the Frue vanners, a very much richer product than 50 per cent. could have been obtained. He endorsed Mr. Collins's statement as to jigging, and considered that the question of the advisability of jigging fine material was one which had not received

sufficient attention. At the Mount Bischoff Mine in Tasmania, 75 per cent. of the total product came from the jigs.

Mr. Alfred James said that he agreed with Mr. Collins that the paper would have been more valuable if the author had given details showing how he had arrived at the figures given in the paper, and he trusted that they would receive some more detailed and complete description after further experiments had been made.

It appeared to him that the last sentence in the paper was pregnant with meaning—in fact, it was the keynote to the whole communication. Mr. Thomas had tried the experiments, and unfortunately had not been able to satisfy himself that he was losing less by the new system than by the old; he simply said the loss was "doubtless less." When they noted the remarks in the paper about the "streamers," it raised the suspicion that much more of the slime tin was being lost than was formerly the case.

They were told years ago that it would be impossible to treat tin by vanners or jiggers, because it was crushed so fine; results showed that this idea arose from lack of knowledge, and that it was possible to treat the fine grains both by jiggers and vanners. It was an accepted fact that for effective treatment by vanners, classification was necessary, and he could not help thinking that at Dolcoath they aimed too much at simplicity. It was impossible to get a machine to give effective work for coarse particles and the finest slimes, and he suggested that if they classified the material first, and had the coarse particles treated on a machine adjusted for coarse particles, and the slimes treated on a machine adjusted to give the best results for slimes, a better result would be obtained, and there would be less loss in the tailings.

He agreed with Mr. Collins as to passing the stuff, which went over the vanners, over frames; he thought it should be done until they were satisfied that they had reached the minimum loss of fine tin.

Mr. S. Herbert Cox thought Mr. James had hardly taken a fair view of the question of treatment, as Mr. Thomas was perfectly clear in his statement that he saved 94 per cent. of the tin on the vanners, and only lost 3 lb. of tin to the ton in passing away from the vanners. He hardly saw how the author's expression of hope that the treatment was satisfactory could do away with the result obtained.

Mr. Franklin White said that it appeared to him to be a mistake to run the pulp over Frue vanners without some attempt

at classification, and considered that better results would have been obtained if something of the kind had been done. The author thought that sufficient pulp was not run over the vanners, and he (the speaker) was of the same opinion, as the stream must be very small. It seemed to him that the percentage of extraction was a little vague—"94 per cent. of the gross contents as ascertained by assays with the vanning shovel"—it would be more useful to know the actual contents, and not that caught by the vanning shovel. As a comparison of systems, it would probably be correct, but for actual results obtained it would be interesting to have some additional data, as they were left in doubt as to how much tin was originally in the ore.

The President pointed out that the vanning shovel was the usual method of assay with tin ores, and as it was used in both the old and new process, it answered very well for the purpose of comparison—it was not a chemical analysis of the tin, but the ordinary Cornish method of assaying.

Mr. White, continuing, said that the results of the experiments which were being made to ascertain whether heavier stamps gave more slimes would be of interest. He thought that probably a larger percentage of slimes would be produced, but unless the weights were carefully gone into, the data would not be as important as they otherwise might.

He thought Mr. Collins was quite right in claiming that water wheels should give the most perfect system of movement if properly run. Cotton spinners used water wheels to a great extent, and in their case it was of the highest importance to have equal running.

Mr. R. G. Elwes observed that there was some ambiguity about the figures in the paper. It was stated that the loss often in the tailings amounted on the average to 3 lb. per ton of stuff and the percentage of extraction shown on the vanners was a little over 94. That looked as if it meant that the tailings ran 3 lb. per ton as they came from the vanners, and in that case it was not a loss, because after coming from the vanners they were re-treated. What they wanted to know was whether the 3 lb. was the final loss after the whole of the treatment, or whether it was what the tailings from the vanners contained before re-treatment; if the latter it was not right to call it a loss.

The President remarked that it was simply the loss from the vanners in the first place, which was still further reduced by the other processes mentioned, and which Mr. Collins thought should be carried further by introducing revolving tables, and dead-frames instead of ordinary buddles.

Mr. D. A. Louis said that at a recent meeting of the Society of Arts, Mr. Collins had expressed the opinion that the old method could scarcely be surpassed, and he was rather surprised to hear him admit that evening that the new method was doing good work. [**Mr. Collins**: If the report is true!] With regard to graduated treatment, he thought there was a good deal to be said in its favour. Mr. Collins had referred to the use of jigs—he had seen the Hart's jigs at work, but they were of a most diminutive character and looked more like models or toys, and he was not surprised to find that they were not doing their duty, and believed that they were ultimately taken away. There was no doubt that a good deal of work ought to be done by jigs, and a considerable amount would also be done with the revolving tables, which in his opinion worked particularly well. He agreed with Mr. Collins that before passing the stuff over the Frue vanners some separation might be done by jigging, and the tailings could be subsequently treated on the revolving tables.

Mr. S. Vivian drew attention to the fact that there were two classes of ore in the Dolcoath Mine, one of which contained a good deal of coarse-grained tin. It struck him that if there was an appreciable amount of such ore some method should be devised for reducing it by rolls, or concentrating by jiggers, or at any rate to crush it by other means than stamps, as a large portion of it went into the slimes, and was probably never recovered.

Mr. Kitto agreed with Mr. Vivian that there were two classes of ore at Dolcoath. In one part of the mine he had seen the tin in very coarse grains disseminated through the rock, while from another part it was so very finely grained that it was impossible to extract it unless it was stamped very finely.

With regard to the President's remarks about the crushing capacity of the plant as compared with some gold mining districts, the rock was a hard tin capel—it was almost as hard as steel, and it was this stuff which contained the tin finely disseminated throughout.

The author spoke of using heavier stamps and it would be interesting to hear the result after they had been given a trial. After the remarks which had been made that evening it was possible they might attempt classification before putting the stuff over the Frue vanners; if so, he hoped the author would communicate the result to the members.

Several speakers had referred to the vanning assay; he doubted very much whether the vanning assay by an old tributer in Cornwall would not be about as accurate as any assay of the tin stuff that could be made.

The President said that in the absence of the author he would endeavour to reply in part to some of the points raised, as he had had an opportunity of looking into the preliminary experiments and also into the present results at the mill.

He agreed with many of the speakers that the figures given in the paper were much less detailed than they should have liked, but he hoped that after the mill had been running a little longer, and accurate figures could be obtained, Mr. Thomas would supplement his paper with further details. The mill had not been running many months, but the time was quite long enough to give the managers a very good chance of estimating the general questions of costs and returns, and he thought it could be taken for granted that their conclusions were not likely to be much out in regard to the preference for one system over another. Whether it was a question of 3 lb. of tin on the average, or 5 lb. of tin according to the grade of the ore, or not, the work had been going on for some 15 months, and whatever imperfections there might be in the system of assays applied equally to the one method of treatment as to the other. Since the operation of the mill there had also been an opportunity of taking the returns as sold to the smelters, as compared with the former returns from the old system, so that he thought they might accept Mr. Thomas's conclusions as correct in a broad sense, whether they might, or might not, have to be modified as regarded percentage of saving on the present system by closer figures at a later date.

Mr. Collins had said that the "streamers" at Dolcoath were satisfied with the change. Seeing that the "streamers" were doing well, and the mine was doing well, everybody was satisfied, which was a very satisfactory state of affairs.

The paragraph in the early part of the paper which stated that "the pulp was now being passed into a classifier" was not quite correct in relation to the present system in use. The intention of the author was to describe first the system formerly in vogue in Cornwall, then the system adopted until lately, and finally the present system.

The question of classification which had been raised by several members was one on which a good deal of argument was possible on both sides. It could, of course, be theoretically shown that classification was essential to good concentration, but it could also be proved that in a number of instances classification had been entirely dispensed with, and treatment on vanners, without classification, adopted in preference. The action of the Frue vanner on classified material was not the same as any other concentrating

machine. Classification was absolutely essential to many machines, but in the case of the Frue vanner it seemed to be absolutely detrimental. Some experiments were conducted by a committee of the Chamber of Mines, at Johannesburg, on the classification before concentration, and treatment without classification, with the result that it was not proved to be an advantage to classify before treatment direct. When the Frue vanner was first introduced into the United States, it was believed that classification was essential, but it had been given up in a large number of the mills in that country. When the first two Frue vanners were put in at the Dolcoath to run against the side inclined belts, the managers were perfectly certain that classification would have to be used, and it was attempted in the same manner as on the two competing machines; the pulp was divided equally between the machines, and it was found that in practice there was no gain by classification, so that theory had, in this instance, to give way to actual facts as established by tests made. His idea was that classification was far better as following Frue vanners than as preceding them; if they took the ordinary pulp as put on a Frue vanner and vanned the tailings, it would be found that the loss occurred in the exceedingly fine mineral—the coarse mineral did not escape—and the percentage of fine material was not greater on the classified than the unclassified. He thought that it was pretty clearly shown that although they claimed to save 94 per cent. at Dolcoath (which seemed high, considering the nature of the tin ore, and the large amount of slimes), the stuff which escaped from the concentrators could, by careful vanning, be made to yield an amount of tin which could be saved by further concentration. He agreed with Mr. Collins that for slimes the ordinary Cornish buddle was not so good as the revolving table, and it was intended to further perfect the extraction at Dolcoath by saving as much of the fine slimes as possible. The tin which escaped was partly in the coarse and partly in the fine; what escaped in the latter could be saved by treatment on buddles or dead frames, and there was no reason why the material should not be treated any number of times, provided it was profitable to do so.

Mr. W. R. Thomas referred to the comparisons between the Cornish and Californian stamps. He (the speaker) did not think anyone could reasonably defend the Cornish, as compared with the Californian, stamp mill. Experienced managers all over the world had condemned the former by adopting the Californian method of cam shafts close to the stems, with revolving tappets to reduce friction, and make the wear and tear on the shoes and dies more

regular. With regard to the use of cast iron, as compared with steel, there was greater economy in using the former when there was a foundry in the neighbourhood, as the broken pieces could be melted down into new shoes and dies. When steel was used, although it was more durable, the refuse could not be utilised on the ground.

Mr. Thomas thought that classification would give a richer product, but he (the speaker) thought that results would not bear out his opinion, as there was no difficulty in getting a clean concentrate from unclassified material, the cleanness depending upon the percentage of sulphur, in the form of mundic, which existed in the ore. Mr. James and Mr. Franklin White both spoke on the subject of classification, but there was no doubt that, in the cases where it had been decided that classification was not an advantage, those who had carried out the experiments were quite aware of the theoretical claims of classification, and had arrived at their conclusions after giving a fair test to both systems.

One of the speakers had referred to the possibility of jigging and coarse crushing to free the tin from the stone; this subject had, however, already been looked into in Cornwall in the past, and any system of graduated crushing would certainly lead to a complication in the machinery which, if the figures as to the present loss were correct, could not be justified by any additional gain. Mr. Kitto had suggested the use of rolls, but it seemed to him that the margin of possible saving, as shown by the present results, could not possibly compensate for the cost of wear and tear and complication of machinery which it involved.

As to the use of the vanning shovel, Mr. Mills, the ore dresser at Dolcoath, was most skilful, and anything that escaped him was not worth bothering about. The Cornish vanner men gave a time for settling in the shovel, and were so expert that no machine could compete against them.

In conclusion, he hoped he should not be taken as being too strong an advocate of the present system; but in the absence of the author he felt justified in replying to the various points raised. Like the other members, he had had to take certain things for granted in Mr. Thomas's paper, but no doubt the author would throw a further light on the subject in his reply to the criticisms which had been made.

CONTRIBUTED REMARKS.

Mr. R. Arthur Thomas. The author is indebted to the President for replying to the various points raised.

It should be clearly understood that the "percentage of extraction of 94 of the gross amount of tin as ascertained by assay with the vanning shovel" is obtained by the vanners only.

Mr. Collins expresses the opinion that the stuff that went over the vanners was still of sufficient importance to require treatment in some other way, and that the revolving tables would be suitable machines, &c. The author certainly never intended to "cast some slight" on the revolving table (buddle), but on the contrary Mr. Collins seems to have confounded the revolving buddle or table with the ordinary buddle. Within the last three years an additional 28 of these tables have been laid down. On reference to the paper the following paragraph will be found:—"The method of dressing adopted is a single treatment on vanners with re-treatment of the tailings after classification, the slimes on revolving buddles (tables), the coarse by concentration and pulverising."

After classification, following the treatment on the vanners, the fines are settled, concentrated on dead frames, again settled and treated on revolving buddles. It is impossible to determine the percentage of extraction in this subsequent treatment, as some of the slimes from the other floors are treated on the same plant.

As this is the first installation of vanners for treating tin stuff in Cornwall, it was thought that the results obtained would be interesting to the members, and no attempt was made to detail the after treatment, as this is so well known. In reply to those who doubted the accuracy of the figures given, the author wishes to state that a great deal of trouble and time has been spent in arriving at the results, which must be taken as correct, and any doubts thrown on their accuracy must equally apply to the manner of determining results of the old method of working with which the new is compared.

A mere reference to the supposed satisfaction of the streamers at the introduction of the vanners is in itself no argument against the results actually obtained. These certainly indicate a high rate of extraction at a very low cost per ton compared with the former system, and the total loss in the whole tailings is doubtless (that is without doubt) less under the new system than with the old.

The amount of coarse-grained tin in Dolcoath is remarkably little. As the mine gets deeper the grain of the tin is finer. For-

merly large quantities of tin stuff containing coarse-grained tin were mined.

The question of classification before treatment on the vanners has already been dealt with by the President, and in the main the author agrees with his expression of opinion as relating to the treatment of Dolcoath tin stuff. In the preliminary experiments it was most certainly thought that classification before treatment on vanners was essential, and the results obtained without classification were as surprising as they were gratifying, hence it was determined to classify after a single treatment on vanners for further dressing by dead frames, revolving buddles, &c.

Subsequent experiments on a larger scale may prove that in some classes of ore it will be an advantage to classify, but for the present class of tin stuff no advantage appears to accrue from so doing.

With regard to the point raised as to assays by the vanning shovel, it should be remembered that no machine or appliance has ever been made yet which will save the quantity of tin which a chemical analysis shows is in the ore, or even give as good results as a vanning shovel in the hands of an expert Cornish vanner.

The vanning assay is essentially a working test, and gives a result which up to the present cannot be equalled by any machines within a commercial limit.

"On Charcoal Precipitation from Auro-Cyanide Solutions."

By JOHN I. LOWLES, Assoc. Inst.M.M.

ONE of the members, in some contributed remarks on Mr. Mactear's paper on the South German Mine, Maldon, asks some questions on the subject of the precipitation of gold from cyanide solutions by means of charcoal. Outside of Victoria it is but seldom employed, but there it is used to the almost total exclusion of any other method. The number of cyanide plants, small and large, throughout the colony is very great, owing to the numerous heaps of tailings on living mines and on abandoned properties, accumulated during the forty odd years of the existence of the quartz-mining industry in Victoria. In the greater number of these installations the charcoal method is employed, in many cases probably in imitation, rather

than with any definite grounds for its superiority over the zinc process.

The methods are, of course, identical down to the extractor house, which in a large plant here, a typical case, contains 198 tubs, each 2 ft. 4. in. high, 2 ft. 1 in. in diameter at the top, and 1 ft. 9 in. at the bottom. In the centre of the bottom of each tub are two small wooden cleats, 4 in. apart, on which rests a glazed drain pipe, 4 in. diameter, flanged end downwards. The centre pipes and tubs are filled with charcoal, coarse for 6 in. from the bottom, and medium size to within 6 in. of the top of the centre pipe, with about 3 in. outside, and 5 in. inside, the tube, of coarse charcoal again. On the top of the charcoal, in the tub itself, *i.e.*, all round the pipe, is a wooden lid, to keep the charcoal from floating and blocking up the outlet pipe.

Another form of filter tub much used has no centre pipe, but there is a false bottom covered with a filter cloth, above which is charcoal, and on top a hinged lid, kept locked, with the feed pipe let into it. The solution leaves the tub under the false bottom by means of a siphon pipe to the next tub.

The preparation of the charcoal is important. Impurities are prejudicial and wasteful. Fine stuff is objectionable, as it will pass through into the sumps with the solutions (and, incidentally, decompose the KCy), besides making a bad filtering medium when closely packed. Heavy compact charcoal, again, is not nearly so efficient as that which is soft and porous. Gold is deposited throughout the whole of light charcoal, so that compact material offers far less surface. Moreover, it would take longer to incinerate. Accordingly the charcoal is ground and sieved to three sizes, coarse (about half-inch), medium, and fine, which last is straightway rejected. The other sizes are thrown into separate tanks of water, and washed; the compact coal sinks, the light, porous stuff of both sizes is skimmed off, and allowed to dry before being used. About 30 per cent. of the original quantity of charcoal is lost by this treatment.

The tubs in these works are arranged in two groups, the first of 144 filters, in 24 sets of six each, for strong solution; and the second of 54 filters, in 18 sets of three each, for alkaline wash (containing 0·04 per cent. KCy) and weak solution.

The solution passing through a 1-in. branch from a 3-in. main, travels down the centre pipe of the top filter in each set, coming up on the outside, over the lid, and through another 1-in. pipe to the second filter, and so on, finally passing away by an iron-lined launder (a tile lining would be better) to its own particular sump, whether alkaline, strong or weak. The top of each of the last five

filters of a set is placed 3 in. lower than the top of the preceding one. The top filter naturally catches most gold, and, when fairly charged, is removed, and each of the remaining five tubs of the set is moved up one step, bringing second filter into first place, a tub with fresh charcoal being put into sixth position. The filter removed is relieved of some of its solution (to save loss during moving), and emptied into a large box with a filter bottom, when what solution there is in the tub drains away to the strong solution sump, leaving the auriferous charcoal ready to be shovelled out and taken to the furnace room. Every day eight top strong solution filters are emptied in this way, so that a tub occupies first position during three days. The flow of solution from the intermediary tanks to the filters is regulated by valves in the main, and an ingenious arrangement prevents that flow being exceeded. At the junctions of the mains with the distributing pipe a small open glass tube is inserted, in which the solution under the right pressure rises to a certain height. A very slight excess over the proper pressure will cause this tube to overflow, obliging the attendant to pay strict attention to the amount of solution he allows to pass through.

The absence of iron in the filter tubs constitutes a slight advantage over zinc boxes, where the iron sieves at any rate do not improve the solutions.

The launders from the filters are connected with all the sumps, so that, if necessary, any one solution can be passed over the whole of the filters, but as a rule the filters are worked in two groups, as shown above, and then about 400 gallons of strong solution per hour pass through 144 tubs, and 300 gallons of weak or alkaline through 54 tubs.

The efficiency of the precipitation of the gold by the charcoal is, of course, an important point. Zinc does not seem to act rapidly enough, or, indeed, quite effectually, on very dilute solutions, whence arose the usefulness of the Siemens-Halske and other electrolytic processes, but with charcoal the strength of solution does not seem to affect the percentage of precipitation at all, which remains uniformly high. At one large works in this colony the solutions going out into the launders from the tubs rarely show more than 3½ grains of gold per ton. Solutions down to 0·04 per cent. KCy are successfully precipitated from at this establishment.

The gold caught by the top filter of a set is considerably purer than that caught in the lower ones. Frequent tests show this to be quite characteristic, steady diminution in the value of the gold taking place downwards through a set, till in the bottom tub it is nearly pure silver. However, as the filters are shifted from the

bottom upwards, they all contain about the same quality gold when burnt.

The reason for this peculiarity is, no doubt, owing to the gold being precipitated first, and other metals afterwards.

No doubt experiments might be usefully made as to the desirability and economy of keeping the contents of the top filter separate from the rest, as containing very pure bullion, requiring little or no refining. With six filters in each strong solution set, the two top filters might be allowed to remain stationary until well charged, the "step-up" process being confined to the lower four.

The exact action of the charcoal has, as far as the writer knows, never been experimentally demonstrated. Dr. Rose, discussing the question in connection with its use in the chlorination process, states that the prevailing opinion is that the hydrogen and hydrocarbons remaining in the charcoal are the active agents in the precipitation. No doubt this would apply also in precipitation from cyanide solutions, the result being the formation of hydrocyanic acid and free gold. This HC_n itself represents a certain amount of loss of an active agent.

It is an accepted fact that the prolonged contact of cyanide solutions with charcoal causes a loss of cyanogen. This undesirable condition obviously obtains, as the same solution (of course strengthened as and when required) circulates for months between the leaching vats and the filters.

Park instances the case of the Kapai-Vermont Works, N.Z., where the ore was kiln-dried before being sent to the pulverisers. Here it paid to employ labour to pick out the lumps of charcoal and partially-carbonised wood rather than sustain the loss of cyanide (and gold also, in the leaching vats), which neglect of this precaution involved.

In the daily trials of the solutions the loss from this cause would not seem to be great. In one case where a 0·25 per cent. strong solution is used, it titrates 0·11 per cent. after use, and the major part of the 0·14 per cent. difference must be accounted for by the gold dissolved.

From the filter-house the auriferous charcoal is conveyed to the furnace room. At the same works, whose extractor plant has been described, there are four reverberatory furnaces, each 10 feet long. The charcoal is charged in through side doors, of which there are six to each furnace, each perforated with a number of holes to admit air to assist in burning off the carbon. The furnaces are arranged in pairs, with a stack to each pair. When charged, a wood fire is lighted in the fire-box until the charcoal is well alight, when the

dampers are closed, and it is allowed to slowly burn itself out. This it does not do completely, so it is sieved in a trommel of 900-mesh, contained in an iron box, what remains in the trommel being returned to the furnace with the next charge. The fine stuff in the box is called "ash." Every 12 lb. of ash carries from 9 to 10 oz. of gold. This ash is mixed with borax (in the proportion of 3 parts borax to 1½ ash, with 1 of dried and washed tailings, when the ash is alkaline), and smelted in graphite crucibles. The pots are filled, and placed in the furnace, the slag skimmed from time to time, and fresh charges of "ash" added, till sufficient gold has accumulated in one pot, or the latter shows signs of wear, when it is poured into a mould. It is certain that some appreciable loss must take place during this operation, owing to the numerous small quantities of valuable ash which are dealt with. Some of the stuff is almost impalpable dust. When charging red-hot crucibles, it is apparent that many fine particles are carried away by the expansion of the air in the neighbourhood of the crucible.

In spite of some very strong points in its favour, it is doubtful if these are sufficient to cause the charcoal method to supersede the better-known zinc process. The writer thinks that the following advantages deserve careful consideration: (1) A high-class bullion is produced. One well-known mine here receives £4 per oz. for its cyanide gold. At any rate the bullion does not have a detrimental constituent, such as zinc, introduced into it by the necessities of the method used, and is of a nature easily refinable. (2) The percentage of the precipitation extraction is high; and this is, as far as the writer knows, regardless of the strength of solution. (3) In spite of the large quantity of auriferous charcoal to be dealt with, a great part of the waste in it is so easily got rid of (by simply letting it burn itself out), and the resultant "ash" is so directly and easily smeltable, that this disadvantage is almost, if not quite, counterbalanced. There is no necessity, as with zinc slimes, for careful oxidising roasts (with some unavoidable loss), or treatment with strong acids.

The one great disadvantage of the method seems to be the greater chance of loss through the immense bulk of stuff handled, and higher cost of the operation itself. Minor points are a somewhat greater outlay for plant and buildings. The effects of the decomposition of the solutions by the charcoal have not yet been very well defined.

At one of the large works here, 700 oz. of gold are deposited per month on 10,000 lb. weight of charcoal. W. M. Davis, in the chlorination process, found that 240 parts of charcoal were required

for the precipitation of 19¼ parts of gold. It will be seen that the proportion is about the same with cyanide solutions.

Theoretically, 1 oz. of zinc should precipitate 6 oz. of gold, but in practice from 4 to 12 oz. of zinc are required for every ounce of gold. Allowing 8 oz. Zn for 1 oz. Au, 350 lb. Zn would be required to precipitate 700 oz. of gold. The cost of charcoal precipitation works out (with charcoal at 9*d.* for 40 lb.) at 4·29*d.* per oz. of gold, or, at these works, 0·545*d.* per ton of stuff treated. Zinc precipitation (with Zn at 4½*d.* per lb.) would cost 2·25*d.* per oz. of gold, or 0·275*d.* per ton of stuff. The labour necessary must also be considered. In these works two men at 7*s.* each per diem are kept constantly employed in the extractor house. This works out at 0·763*d.* per ton of stuff treated. With zinc extractors the work could be done by the foreman, or even allowing one man at 7*s.* the cost would be reduced by half, that is, to 0·382*d.* per ton.

To recover the gold from the charcoal "ash" costs at these works 1*s.* per oz. of bullion 900 fine, exclusive of labour. This is accounted for by the large quantity of stuff to be smelted, with a consequent large expense for fluxes, fuel, and pots. In some works in New Zealand and South Africa, the cost of the recovery of the gold from zinc slimes is as low as 2*d.* per oz., though probably it would not average more than 800 fine.

This particular item does not of course make much difference when spread over a number of tons of low-grade ore, for the total cost at this particular plant, with cyanide at 1*s.* 2*d.* per lb., is only 2*s.* 3*d.* per ton.

Considering these works particularly with reference to loss through the large bulk of stuff frequently handled, we see 10,000 lb. of charcoal transferred from 208 filters (*i.e.*, eight filters per day twenty-six days a month) to the draining box, divided among four furnaces for incinerating, and finally smelted in charges of not less than 12 to 15 lb.

There remain the questions of increased outlay for plant and buildings, but these are not very serious.

Still considering the same works, each filter tub costs 25*s.* complete, so that the whole 198 cost £247 10*s.* Zinc extractors of the necessary capacity, with trays, steel side launders, &c., cost £60.

Some expense on this head might be saved by the substitution of large shallow wooden tanks, such as are in use at the Mount Morgan chlorination works, though perhaps the thorough contact between the solution and the charcoal which is most favourable to successful precipitation would not take place in them as well as in the numerous small tubs.

The area of the building occupied by the filters and connections is fully 1,000 sq. ft. Zinc boxes would require but a small portion of this.

There are many members in Australia no doubt who could have written much more fully on this subject, but the writer will have achieved his object if he be successful in causing a discussion on the merits of the charcoal method amongst some of the many prominent cyanide members of the Institution.

DISCUSSION.

Mr. Alfred James said that the paper was written in such a frank manner as to almost disarm criticism. A number of objections immediately forced themseives to the front, but the author admitted them, and stated that he scarcely thought that it was likely to supersede the better known zinc process. For effective precipitation, the solution required to be in a very favourable condition, and the rate of flow through the extractor boxes had to be very carefully regulated. In the case in question, if the solutions were allowed to flow through at the rate of 760 gals. a hour, instead of 400 gals. the effluent rose from 3½ to 10 grains of gold per ton, which indicated the absolute care required in the regulation.

In referring to the disadvantage of zinc boxes, the author had apparently overlooked the fact that the iron sieves had no chance of contaminating the solution when zinc was present, as zinc was so electro-positive to iron.

He was much struck with the unhandy and unwieldy amount of precipitant necessary to obtain effective results. For a small plant treating about 2,400 tons a month, 198 tubs and four furnaces to oxidize the charcoal were required. If the size of those South African plants, which were about ten times as large, were taken into consideration, there would be nearly 2,000 tubs and 40 furnaces required. In the plant in question, one ton of stuff had to be fused in small crucibles in 14 lb. charges; in South African practice 10 tons of stuff would have to be fused, and it was so exceedingly light that the loss must be very heavy.

As showing the difference between the use of charcoal and zinc the following figures were of interest:—The 144 tubs contained 720 cub. ft. of charcoal; the work could be done quite as effectively with 45 cub. ft. of zinc. The 300 gallons of weak solution had to pass through 270 (?) cub. ft. of charcoal, 34 cub.

ft. of zinc would do the work quite well. Material containing under 5½ per cent. of gold had to be melted in crucibles, whereas slimes with acid treatment contained as much as 30 per cent. In the treatment of slimes, borax, which had to be dehydrated, was a most expensive item, and by the charcoal process no less than 24 times more borax was used than in the case of zinc.

Mr. Mactear asked if Mr. James could give them some idea of the relative value of the gold. The cost of refining the zinc bullion was considerable, and that cost would be saved by the process in question.

Mr. James stated that he understood the cost of refining bullion 800 fine, was 4*d.* per oz.

Mr. H. L. Sulman drew attention to the contradictory statements in the paper with regard to the destruction of cyanide. The author quoted Park as to the advisability of keeping a man to pick out the charcoal from the ore, the neglect of which precaution involved a loss of cyanide; Mr. Lowles also indicated that the presence of finely divided charcoal occasioned cyanide losses. He (the speaker) had that afternoon passed a solution of cyanide of potassium of 0·1 strength through charcoal prepared as nearly as possible in accordance with the directions given, and after passing through a column of some 10 in., about three times, the cyanide was reduced from 0·1 to 0·07, which was a considerable loss and tended to show that precipitation of gold by charcoal certainly did give rise to very considerable cyanide losses—a point which Mr. Lowles had apparently overlooked in his paper.

Dr. Rose pointed out that there was a small arithmetical error in regard to the quantity of charcoal required to precipitate the gold. Mr. Lowles stated that 700 oz. of gold were deposited per month on 10,000 lb. weight of charcoal, which was something like 300 to 1 and not, as Mr. Lowles put it, about 12 to 1.

Mr. Picard thought the paper a very useful one. It appeared to him that the rate of flow was remarkably slow, it worked out to about 17 oz. per hour per set of six filters, which would be out of the question in a large plant. He was of opinion that zinc was a far better precipitant than charcoal, and, with a plant of the size mentioned in the paper, would give just as good results. The author said that the strength of the cyanide was of no importance; he thought it was, with a big enough plant.

The clean-up was a very expensive and elaborate process, and he would sooner clean up zinc stuff than the ash mentioned.

He would like to know what the loss was. The author should be able to give some figure as he must know the liquor assays, or the

assay of the tailings going into the vats, and also as they left the vats. It ought to be possible to get very closely to the clean up loss, which he should not be suprised to see come out at 5 per cent. The author stated that the clean-up of the zinc process cost 2*d*. per oz.; he should like to know where that was done.

Mr. James: at the May Consolidated.

Mr. Picard, continuing, said Mr. James had estimated the amount of ore treated in the plant in question, and he thought it would be interesting to have the exact figures.

He was sorry no one had gone into the theory of precipitation by charcoal, and suggested that an electrolytic process might be an advantage.

Mr. James remarked that he understood that the most important mines in Africa had done away with the electrolytic process, and had reverted to the use of zinc.

The President thought that the discussion had shown clearly that the process was not all that could be desired. The difficulty in handling such extremely fine ash, and the care necessary in burning off the charcoal in such large quantities were serious objections to its use, and doubtless people who had not adopted the process had good reasons for not doing so.

"Notes on the Electro-Deposition of Vanadium."

By SHERARD COWPER-COLES, M.Inst.M.M.

VANADIUM has not received the attention it deserves at the hands of electro-metallurgists. Vanadium oxidises slowly in the air, and has a very high melting point, about 2,000° C. (equal to 3,600° F.), and is neither volatile nor fusible when heated to redness in hydrogen. Vanadium is widely distributed,* but it is not often found in large quantities. It was first discovered by Seftström in 1830, in the iron obtained from the Taberg ores.

The metal is not readily attacked by hydrochloric acid, either when hot or cold, neither strong nor dilute sulphuric acid act upon the metal in the cold, but when heated with strong acid the metal slowly dissolves, giving a greyish-yellowish solution. Both hot and cold solutions of caustic soda are without action on the metal, but

* See *Mineral Industry*, vol. vi.

when fused with the hydroxide, hydrogen is evolved, and vanadate formed. Nitric acid of all strengths oxidises the metal with violence, evolving nitrous fumes and forming a blue liquid. The specific gravity of vanadium at 50° F. is 5·5, being somewhat lighter than zinc, and about twice as heavy as aluminium.

Metallic vanadium made by reduction from the dichloride in hydrogen, is a light whitish-grey coloured powder, which under the microscope reflects light most powerfully, and appears as a brilliant crystalline mass, possessing a silver-white lustre.*

Vanadium was originally found in Mexico. Recently it has been discovered in considerable quantities near Santa Márta in Spain. In the Spanish mines, it is found associated with lead, tellurium, gold, and silver. Some of the ore contains as much as 124 lbs. of metallic vanadium to the ton.

Vanadium has also been mined on the elevated plains of the Great Chain of the Andes, where anthracite mines exist, containing two parallel inclined beds of vanadium-containing coal, from 2 to 3 metres thick and 1,004 metres long. It is also found in Peru, in certain anthracite coals, which, according to analysis, contain about 4·5 per cent.

An interesting paper has recently been read before the Geological Society of Washington by Mr. W. F. Hillibrand, on the distribution and quantitative occurrence of vanadium in the rocks of the United States.

Gore has endeavoured to deposit vanadium from aqueous solutions, but with unsatisfactory results. The electrolyte employed by him was one composed of vanadic acid, dissolved in pure dilute hydrochloric acid by means of an electric current, using a retort carbon anode and a platinum cathode; the gas set free at the anode was observed to have a smell of ozone. He also tried a solution of dilute sulphuric acid with pure vanadate of ammonia, and electrolysed the solution with platinum electrodes. The electrolyte turned a bluish-black colour at the cathode, and a jet-black powder of some thickness was deposited upon it.

Schicht dissolved vanadium chloride in water containing hydrochloric acid, and electrolysed the solution. Inert deposition took place in the blue liquid, the vanadium acid being merely reduced to oxide.

The author has succeeded in obtaining brilliant metallic deposits of vanadium, the colour being almost as white as that of silver, from a solution prepared as follows:—

* Roscoe and Schorlemmer's *Treatise on Chemistry*.

The vanadic anhydride, V_2O_5, was boiled with an excess of caustic soda, the sodium vanadate thus formed being decomposed by an excess of hydrochloric acid. The proportions were as follows:—1·75 parts of vanadic anhydride (equal to 1 oz. per gallon of solution) were dissolved in 2 parts of caustic soda and 160 parts of water, to which 32 parts of hydrochloric acid was afterwards added.

The best results were obtained with a current density of 18 to 20 ampères per sq. ft., the E.M.F. at the terminals of the electrolysing cell being 1·88. The solution was worked at a temperature of about 180° F. If higher or lower current densities were used red-brown oxide was deposited along with the metallic vanadium. A reddish-brown non-adherent powder was deposited from cold solutions at the same current density. The solution when first made up was of a light greenish-yellow colour. After boiling with a carbon anode for some time it became darker, and after passing a current it turned a rich dark green, when, a good deal of metal having been taken from the solution, it appeared bluish. When a solution containing 2 oz. of vanadium to the gallon was electrolysed the deposit was not so white as that obtained from a solution containing 1 oz., the deposit having a steely appearance.

Vanadium is found to increase the tensile strength of iron, copper, and aluminium when added to those metals, also the ductility. The present use of the metal is almost exclusively confined to converting aniline into fixed black dyes for indelible ink, and the colouring of glass.

The present price of vanadium oxide is about £2 per lb. Sir Henry Roscoe some years ago produced a few grams of metallic vanadium. Now that the author has shown that it can be readily electro-deposited in a metallic form, it is to be hoped that new uses will be found for this interesting metal.

EIGHTH ANNUAL GENERAL MEETING, 29th March, 1899.

Mr. WALTER McDERMOTT (*President*) in the Chair.

The Secretary having read the Minutes of the last Annual General Meeting, which were duly confirmed, Mr. Franklin White and Mr. H. D. Griffiths were appointed scrutineers of the balloting papers for the election of the members of the Council.

The Report of the Council for the Session 1898–9 and Balance Sheet for the year ended 28th February, 1899, were next submitted, and on the motion of **The President**, seconded by **Mr. C. A. Moreing**, was unanimously adopted.

REPORT OF THE COUNCIL.—SESSION 1898–9.

The past year has again been a successful one for the Institution, both as regards the work accomplished and the continued increase in membership.

The present strength of the Institution is as follows:—

Honorary Members	4
Members	308
Associates	137
Students	136
Total	585

Taking the preceding four years the respective totals were: for 1894–5, 295; for 1895–6, 372; for 1896–7, 449; and for 1897–8, 501. It will be seen that the increase in membership during the past year has been greater than that for any of the four years mentioned, and this is a source of much satisfaction to the Council, as evidence of a still increasing appreciation of the value of the Institution by those following the mining and metallurgical professions.

The Council has with regret to mention the loss by death of Messrs C. J. Smith, A.R.S.M., and E. Y. Surtees.

The following gentlemen have been elected during the year just ended :—

Members :—W. A. Akers, D. P. Brown, G. F. Beardsley, A. A. Blow, T. Bowyer-Bower, A. R. Browne, Ross E. Browne, Matthew T. Brown, J. S. Cherry, Jas. Colquhoun, Geo. Chalmers, P. E. O. Carr, T. J. Greenway, H. H. Greenway, A. P. Griffiths, J. A. Gilmour, Henry Hooke, E. W. Hawker, Ben. Howe, C. F. Heathcote, W. Pellew-Harvey, T. W. Jenkins, D. S. Johnston, Henry A. Judd, R. B. Johnston, J. H. Cordner-James, Hans Knutsen, E. Lidgey, J. Morrish, Hugh McPhee, G. W. Maynard, John McConnell, J. T. Milligan, Wm. Morgans, B. A. Nathan, W. J. Newall, E. Naumann, Joseph Pryor, H. M. Porter, H. A. Pringle, Thos. Pascoe, R. M. Raymond, Joseph Richards, L. A. Scowden, W. H. Skockley, W. van Slooten, D. T. Thomas, Karl Vogelsang, W. S. Welton, G. A. Wright, Gardner F. Williams.

Associates :—H. T. Butcher, G. L. Burnett, M. H. Burnham, W. J. Hill-Cathrine, Clement Dixon, J. T. Dixon, J. G. Hardy, W. Hopwood, C. S. Herzig, T. H. Lee, C. R. Pinder, C. E. Parsons, A. J. Rickard, Paul de Rilly, A. Richardson, A. H. Turton, J. Watts, H. E. West, Frank Western.

Students :—L. J. Abrahams, Allan Archer, C. E. Atkinson, C. J. Bagenal, H. B. Bateman, G. Noel Brown, W. H. Branscombe, H. D. Crowder, J. A. Cameron, Gordon Duff, K. H. Daniel, J. Duncanson, Alfred Fox, R. Farina, W. Gau, C. M. Harris, Wm. Hamilton, S. R. Jameson, O. Jones, A. Kaye, C. N. Keith, R. Krall, N. Lynn-Smart, H. S. Morris, T. G Madgwick, A. N. Machay, C. R. Morrison, E. McGowan, S. Nuthall, E. H. Pullinger, R. Rhodes, John Roberts, P. C. A. Stewart, R. B. Smith, H. B. Williams, C. F. Witty, M. S. Weston, D. Watts.

During the year the papers read and discussed at the ordinary meetings have been fully up to the average of previous years in value and interest, and a number of them have given rise to very full and useful discussion. The titles of the papers read are given below; and, in addition, a list accompanies this report which contains the titles of all the papers which have appeared in the Transactions of the Institution.

"The Minerals which accompany Gold, and their bearing upon the Richness of Ore Deposits," by T. A. RICKARD, M.Inst.M.M.

"Notes on the Electrolysis of Argentiferous Copper," by WILLIAM TERRILL, M.Inst.M.M.

"Some Notes on the Recovery of Gold from Cyanide Solutions," by SHERARD COWPER-COLES, Assoc. Inst.M.M.

"Determination of the Present Value of a Mine on the Rand," by FRED. HELLMANN, M.Inst.M.M.

"Notes on a Combination Mill in the United States," communicated by WALTER MCDERMOTT (*President*).

"Filter-Press Treatment of Gold Ore Slimes (Hannan's, West Australia)," by WILLIAM MCNEILL, M.Inst.M.M.

"The Portable Mono-rail Transport System," by C. C. HOYER MILLAR, Assoc. Inst.M.M.

"On the Occurrence of Diamonds at Inverell, New South Wales," by H. M. PORTER, M.Inst.M.M.

"On the use of Fuel at Caylloma, Peru," by BERNARD HUNT, Assoc. Inst.M.M.

"The Treatment of Broken Hill Sulphide Ores by Wet Extraction Processes, and the Electrolytic Deposition of Zinc," by EDGAR A. ASHCROFT, M.Inst.M.M.

"Notes on the Payable Conglomerate Beds encased in Sandstone, and the Mining Methods in use for their Extraction, on the Witwatersrand Goldfields, South African Republic," by A. F. CROSSE, M.Inst.M.M.

"Notes upon Mining and Concentration of Copper Ores in the Lake Superior Region," by P. R. ROBERT, M.Inst.M.M.

"Notes on Lode Tin Mining in the Malay Peninsula," by W. H. DERRICK, Assoc. Inst.M.M.

"Notes on Dry and Wet Crushing with Cyanide Treatment, in New Zealand," by JOHN MCCONNELL, M.Inst.M.M.

"Notes on Dry and Wet Crushing," by FRANK MERRICKS, M.Inst.M.M.

"Notes on a Process for Treating Slimes without Filtration or Decantation," communicated by ALFRED JAMES, M.Inst.M.M.

"Notes on Fouché's Aëro-Condenser," by Dr. A. L. SIMON, M.Inst.M.M.

"The Tin Mines of Bolivia," communicated by WALTER MCDERMOTT (*President*).

"Minerals found in the Silver Lodes of Tatasi and Portugalete, Bolivia," by MALCOLM ROBERTS, Assoc. Inst.M.M.

"The Deep Alluvial Leads of Victoria, Australia," by ERNEST LIDGEY, M.Inst.M.M.

INSTITUTION OF MINING AND METALLURGY.

Balance Sheet 28th February, 1899.

Dr.

	£	s.	d.	£	s.	d.
To Sundry Creditors				233	5	1
Subscriptions paid in advance				29	3	3
Accumulated Fund—	£	s.	d.			
Balance at 28th February, 1898	690	11	11			
Add						
Amount over-estimated for Sundry Creditors 28th February, 1898	30	0	0			
	720	11	11			
Add Excess of Income over Expenditure for the year ended 28th February, 1899, as per account (annexed)	20	11	6			
				741	3	5
				£1,003	11	9

Cr.

	£	s.	d.	£	s.	d.
By Cash—						
At Bank, Messrs. Prescott, Dimsdale, Cave, Tugwell, and Co., Limited—	£	s.	d.			
On Deposit Account	100	0	0			
On Current Account	196	18	6			
				296	18	6
In Secretary's hands				2	0	1
						298 18 7
Investment—						
£502 London and South Western Railway 3 % Consolidated Debenture Stock (at cost)						599 10 9
Interest accrued						2 3 11
Office Furniture, &c.						78 6 0
Library						24 12 6
						£1,003 11 9

Dr. Income and Expenditure Account for the Year ended 28th February, 1899. Cr.

Expenditure.

	£	s.	d.	£	s.	d.
To General Expenses,—viz.:						
Salaries	182	0	0			
Rent of Offices	112	8	4			
Office Cleaning	11	18	4			
Coals and Gas	1	12	0			
Stationery and Printing	37	5	0			
Postage and Telegrams	86	16	9			
Travelling Expenses	5	9	5			
Hire of Theatre for Meetings	15	19	6			
Reporting	17	16	2			
Printing *Transactions*	308	5	9			
Bank Charges	0	5	1			
Audit Fee	10	10	0			
Rent of Telephone	20	0	0			
Electric Lighting	0	9	5			
Sundry Expenses, including Extra Assistance	29	10	1			
				840	5	10
To Balance carried to Accumulated Fund—						
Excess of Income over Expenditure for the Year ended 28th February, 1899				20	11	6
				£860	17	4

Income.

	£	s.	d.	£	s.	d.	£	s.	d.
By Subscriptions—									
Life Members	21	0	0						
Members	293	9	5						
Associates	105	5	9						
Students	79	15	0						
Subscriber	1	1	0						
				500	11	2			
By Subscriptions—									
For 1893 received in 1898	1	1	0						
1894 do. do.	1	1	0						
1895 do. do.	1	1	0						
1896 do. do.	17	6	6						
1897 do. do.	193	14	6						
				214	4	0			
By Entrance Fees				113	8	6			
By Dividends on Investment				14	11	0			
By Interest on Deposit				1	13	8			
By Sale of *Transactions*, Reprints, and Binding				16	9	0			
							860	17	4
							£860	17	4

ARTHUR C. CLAUDET,
Hon. Treasurer.

Examined and found correct, WOODTHORPE, BEVAN, & CO.,
Chartered Accountants, *Auditors.*

Leadenhall Buildings, London, E.C.,
March, 1899.

"Notes on Blue Asbestos," by H. F. OLDS, Assoc. Inst.M.M.

"Notes on Dry Crushing and Cyaniding of Rand Ore," by FRANKLIN WHITE, M.Inst.M.M.

"Notes on Economical Mining at the Myalls United Gold Mine, New South Wales," by WM. R. THOMAS, Assoc. Inst.M.M.

"Mica Mining in Bengal, India," by A. MERVYN SMITH, M.Inst.M.M.

"On Crushing and Concentration at Dolcoath Mine, Cornwall," by R. ARTHUR THOMAS, M.Inst.M.M.

"On Charcoal Precipitation from Auro-Cyanide Solutions," by JOHN I. LOWLES, Assoc. Inst.M.M.

"Notes on the Electro-Deposition of Vanadium," by SHERARD COWPER-COLES, M.Inst.M.M.

A Balance Sheet and statement of Income and Expenditure for the year is appended, and the Council have pleasure in calling attention to the very satisfactory financial position of the Institution.

The Council has steadily kept in view, in its election of new members and associates, the provisions of the bye-laws which specify certain qualifications as essential to election; and it is the observance of these provisions which gives a special value to membership in the Institution.

The annual subscriptions amount to about £940. The total expenditure and liabilities shown by the attached statement is £840 5*s.* 10*d.* This shows an excess of regular income over expenditure of about £100 for the year, in addition to the sum of £147 for entrance fees received or still due. The invested funds are £599 10*s.* 9*d.*, which can now be further increased.

During the year the office of the Institution has been removed to larger and more commodious rooms in the same building, and a reading room for members is now available.

WALTER McDERMOTT, *President.*
S. HERBERT COX,
BENEDICT KITTO,
ARTHUR L. PEARSE, } *Hon. Secretaries.*

The President announced that the scrutineers had reported that the balloting papers were in order, and that the following gentlemen constituted the new Council:—

COUNCIL—1899-1900.

President:

S. Herbert Cox.

Vice-Presidents:

C. Algernon Morring.
Edgar Taylor.

Members:

Charles J. Alford.
R. T. Bayliss.
Chas. Butters.
A. G. Charleton.
Arthur C. Claudet.
William Frecheville.
J. Hays Hammond.
Edward Hooper.
Alfred James.
Hennen Jennings.
Benedict Kitto.
Bedford McNeill.
Arthur L. Pearse.
H. C. Perkins.
Edward Riley.
T. Kirke Rose.
R. P. Rothwell.
William Thomas.
R. A. Varden.

Hon. Secretaries:

A. G. Charleton.
Benedict Kitto.
Arthur L. Pearse.

Hon. Treasurer:

Arthur C. Claudet.

The retiring President then vacated the Chair in favour of Herbert Cox, and the proceedings of the Annual General M terminated.

SEVENTH ORDINARY MEETING, 19th April, 1899.

Mr. S. HERBERT COX (*President*) in the Chair.

The Minutes of the previous meeting having been read and confirmed,

Mr. S. Herbert Cox delivered the following Presidential Address:—

PRESIDENTIAL ADDRESS.

By S. HERBERT COX (*President*).

GENTLEMEN,

In taking the chair for the first time as President of the Institution of Mining and Metallurgy, I must express the honour I feel that my election by your Council has conferred upon me. I may at the same time state that I was quite unprepared to be nominated this year, and would, consequently, ask your indulgence if, as I fear will be the case, my address this evening falls short of the standard you have a right to expect from anyone who assumes the high functions of this office.

I feel especially that, following our late President, I shall have a difficult position to fill.

However, I trust to devote my best energies to the work, and rely upon members not being too critical of any shortcomings on my part which may become evident as the year goes on.

The first point to which I would call your attention is the present position of the Institution. You will find from the report of the Council that no falling off has occurred during the past year, but rather that our strength is increasing both numerically and financially. As regards members, we have now 4 honorary members, 308 members, 137 associates, and 136 students, an increase for the past year of 84 in all.

You are aware of the care that is being exercised by your Council in the election of new members, and will readily understand that

this being the case, there is a steady and constantly increasing demand, on the part of those qualified for membership, to join the Institution. I hope that in the not distant future all mining and metallurgical engineers of standing throughout the world will have become enrolled upon our list of members.

We are greatly indebted to our honorary treasurer, Mr. Claudet, for his untiring attention to the affairs of the Institution. Not only does he look after our finances—no unimportant matter in itself—but he also occupies the editorial chair, and the *Transactions* published are I am sure a great credit to the Institution, their excellence being the best gauge of the amount of time and patience devoted to them. Our present financial position, with which I think you have cause to be satisfied, was explained to you at the Annual General Meeting last month, and for so young an Institution is almost unique.

During the past year it has been decided by the Council that any gentlemen who are not eligible under the rules for membership but who wish to acquire the *Transactions* of the Institution, shall be allowed to do so by the payment of an annual subscription of one guinea, for which sum they have also the privilege of attending our meetings, and thus it is hoped the words of wisdom communicated by you and printed in the *Transactions* will be more widely circulated than heretofore.

The papers which have been read before the Institution during the last 12 months are 27 in number, of which four have been communicated, eight are by associates, and the remainder by members of the Institution. Their scope is very wide, embracing matters of mineralogical, metallurgical, and mining interest, and in some cases they have furnished us with figures showing the actual cost of the operations which are described. These papers have received your attention on the evenings when they were read, and in the majority of cases have given rise to discussions which show the interest they excited, so that it is unnecessary for me to refer to any of them in detail.

I would, however, like to say that the full and complete information given by Mr. Ashcroft regarding the treatment of the Broken Hill sulphides by the Ashcroft process is of particular interest to a body such as ours, more especially because it is not always the case that an author has the pluck to record the history of a failure, however instructive it may be. Other, and not perhaps less important, papers have given us records regarding crushing and cyaniding under different conditions, various methods of slime treatment and concentration, and the distribution and modes of occur-

rence of metals and minerals, such as gold and tinstone, all of which subjects merit our careful attention.

Important as the papers are, the discussions are frequently of quite as great importance, and these may be rendered more and more valuable if our members attend the meetings regularly and having read the papers beforehand, as they always have the opportunity of doing, will give us the benefit of their experiences on the matters therein contained.

I think that so far as metalliferous mining is concerned the year 1898 has been specially notable for the expansion that has taken place in the gold mining industry. Wherever we turn the gold yields have been important, but the Rand and Western Australia have, without doubt, far surpassed other localities in the advances which they have made, while Klondyke has also had a large output during the past year, although entirely from alluvial gold.

During 1898 the total increase in the output of gold for the world has been 2,405,932 oz., according to the *Engineering and Mining Journal* of January 7, the figures being as follows:—

	Oz.
Total gold output for the world during 1898	13,805,407
" " " " 1897	11,399,475
Increase during 1898	2,405,932

The most important fields have been—

South African Republic	4,555,009
Australasia	3,538,370
United States	3,110,788
Russia	1,216,100
Canada	686,502
India	369,018
Mexico	365,032
China	321,296

some of the figure being only approximate. The balance is accounted for by a large number of smaller producers.

There have naturally been new processes introduced during the year for the extraction of metals from their ores, but I hardly feel that one is justified in either applauding or criticising them here. Every process may be of value for one particular class of ore, and specially applicable to one mine or group of mines, but no process can exist which is capable of universal application.

Holding this view, with which I am sure every member of the

Institution agrees, I think I can hardly do better in the time at my disposal than direct your attention to some of the conditions which call for fulfilment in order to render mining a successful industry.

The profession of a mining engineer has materially advanced during the last 25 years, and while mining to-day has by no means lost its speculative element, it has under more and more scientific methods assumed a position which it has probably never occupied before, at any rate during the present generation. Not many years ago business men would have considered that holding shares in mines was a rather more pronounced form of gambling than betting on horse-racing. At present one seldom meets anyone who is not interested to some extent in mining, and while the ventures of certain persons have resulted in the acquisition of wealth and much self-appreciation of their own ability as financiers; others have lost money and at the same time have had their belief in their genius for finance rudely shaken. It is the principal duty of a mining engineer to further establish mining as an industry and to eliminate the speculative element as much as possible.

I would like to ask the indulgence of the members of the Institution, whilst I make a slight digression in order to point out a few ways in which the advance of mining has affected the general appreciation in which it is held and has induced not only capitalists but even governments to undertake large and costly works with the view of fostering the mining industry.

It was not in older established countries that this assistance originated, but mainly in the Colonies and America, where the advantages of the art of mining and the results which accrue therefrom have been most fully appreciated. In each of the Australian Colonies government assistance has enabled many mining districts to be worked which without this assistance would probably still be unremunerative.

In the early days of mining in Victoria, when alluvial gold was first discovered, the cost of cartage to the mines was £80 per ton. The fabulous riches found enabled the miners to contend against every disadvantage for a time, and as the richer gravels became exhausted the facilities of communication increased and the costs of living were reduced. Railways were eventually built and now after all the easily accessible gold has been won, the increased facilities of transport have rendered it possible to work at a profit quartz of very low grade. The same has been the case in New South Wales and Queensland, where the gold mining industry has been fostered with the best results. In New South Wales again the Government has expended large sums in improving the Newcastle harbour and pro-

viding wharves, hydraulic cranes, and railways for the use of the collieries; and South Australia and Western Australia have not been behindhand in expending public money with the object of rendering their mineral deposits reproductive.

In Western Australia, had it not been for the enterprise of the government in railway construction, it is very doubtful whether the magnificent mines that are now being worked at Kalgoorlie would be more than bare prospects, and the Colony has not stopped short, having provided the goldfields with railway communication, but is now proceeding with a scheme for pumping fresh water from the Darling Range for purposes of distribution to the fields.

New Zealand again has not hesitated to assist mining in every possible way. It is true that no great lengths of railway have been laid down to the mining fields, but much money has been expended on improving the harbours of Westport and Greymouth, where indeed short lines have been constructed to serve the collieries. Large water races and sludge channels have also been built and maintained, and have enabled alluvial deposits to be worked which could not otherwise have been dealt with, and monetary assistance has frequently been given to prospectors, whilst sinking to deep levels and pumping have frequently been subsidised by the government.

The British South Africa Company has built a railway to Bulawayo, and is preparing to continue it through Gwelo to connect with the Beira and Salisbury Line, and this would appear to be almost entirely in the interest of the mining industry. It is true that there may be trifling Imperial reasons for the construction of these lines with the object of connecting Cairo with the Cape—and it may also be true that agricultural produce and cattle will be transported over them—but after all the main interest in Rhodesia at present centres in mining, and were it not for that the lines would never have been built. Even in such inhospitable districts as Klondyke we find companies building railways and providing transport from the coast, no doubt with the idea of gain, but gain to be derived from serving the mines there. In a word, a due appreciation of the value of mines and a belief in their permanence has superseded the indifference—to say the least of it—with which they were regarded less than half a century ago.

To return—you will appreciate the fact that the speculative element in mining can never be entirely eliminated, and indeed I hardly think that it would be to the interest of mining shareholders that it should be, but mining engineers should strive gradually to raise a mine from the speculative period in its history to that stage

at which regular dividends can be made and foreseen with some certainty for years ahead. I shall recur to this subject later on this evening, but I would point out to you now that while in gold mining we are certain that no fluctuations will take place in the value of the gold, in every other form of metalliferous mining the fluctuations in value of the metal alone will always afford a sufficient reason for considerable variations in the value of a mining property from year to year.

I am pleased to say that at the present time these variations in value of metals are mostly in favour of the mines, with the exception of silver, which continues to be quoted at an abnormally low price.

In August, 1897, silver was quoted as low as 24·93*d.* per oz., having fallen from 29·74*d.* in January of that year. It recovered to 26·29*d.* per oz. in January, 1898, and with minor fluctuations during the year was at 27·45*d.* per oz. last December, and the latest quotation is 27½*d.* per oz.

The price of silver, however, is certain to appreciate in time with the increased production of gold, and I must remind you again that the world's output of gold for 1898 exceeded that of 1897 by nearly two and a half millions of ounces or approximately £10,000,000 sterling.

As regards other important metals the prices are good. Copper is quoted at £71 12*s.* 6*d.* per ton and tin at £109 15*s.* per ton, prices which have not been reached for many years. Iron again is unusually high in value now; spelter is quoted at £27 15*s.* per ton; and lead at £14 10*s.* per ton.

As a comparison of the fluctuation in prices during last year the following figures are instructive:—

	January, 1898.			January, 1899.		
	£	*s.*	*d.*	£	*s.*	*d.*
Copper	48	12	6	59	3	9
Tin	63	0	0	92	5	0
Lead	12	11	3	13	10	0
Spelter	18	1	3	24	7	6

These prices are all in favour of the mines and, incidentally, of the mining engineer, for they enable him to make profits out of mines which, when the prices are lower, would not repay the cost of working; but in this connection I feel it very essential that some remarks should be made regarding the working of mines and the records which are, and should be, kept of the costs and methods of working.

I remember hearing an observation made in this room to the effect that someone did not want to have anything to do with a mine that would not stand mismanagement, and I suppose every one of us would re-echo this wish. Unfortunately, however, we cannot always pick and choose, and I fear that few of us have had the good fortune to be always connected with mines of the better sort. The old Cornish proverb that "a good bal makes a good captain," so often quoted, is still true, but the best management is often to be found in the lowest grade mines. A capable mine manager of to-day requires to be a man of parts. The days when it was considered that an illiterate man, whose whole knowledge consisted of an appreciation of the price at which a level should be driven, ground stoped, or a shaft sunk, coupled with an ability to do the manual work himself as well as or better than his men, was the best manager for a mine, have passed away for ever, and although we still occasionally hear the "practical man" quoted, no one in his senses would employ such a man now-a-days in preference to a properly trained manager. A practical man is still required at the head of affairs, but he must have a wide grasp. He must of course have a good knowledge of mining as a first consideration, but he equally ought to possess a considerable acquaintance with the treatment of ores and a fair knowledge of chemistry, so as to be able to investigate the causes of unexpected difficulties in treatment which may present themselves. He should be an accountant, a mechanical engineer, and an electrician, and able, moreover, to handle men and get a fair day's work done for a fair day's pay; I might add that he must be a diplomatist and more or less of a bush lawyer in order to deal with those with whom he is brought in contact outside the actual working of the mine; and there may be other qualifications I have not mentioned. Very few men possess all these attributes in perfection, and perhaps the most necessary qualification that a manager requires is a knowledge of his weak points, and an ability to select capable and reliable men for those posts with the details of which he is least aquainted.

If we were to review all the mines of the world I question whether 1 per cent. of them would stand mismanagement, and I venture to say that a considerably larger percentage suffer from that complaint. Nearly all mines require everything to be done that can be done to reduce the cost of working to a minimum and to secure the best returns from the stone available for treatment, if they are to be worked at a profit.

We are all acquainted with instances in which good profits have been realised in gold mines which did not yield more than 2 dwt.

of gold per ton where all the conditions have been favourable, and I regret to say there are other cases where a yield of 1 oz. per ton has not been remunerative. It seems a marvellous thing that these variations should occur, for a difference between 8*s*. and £4 per ton in the cost of working would seem to be almost impossible.

What applies to gold applies equally well to other ores, and the working costs vary almost indefinitely at different mines. I fully admit that the systems of capitalisation that are adopted are not always the best, and that if profits are made, there are cases in which an undue proportion of them are absorbed by the paid-up shares; but this question can have no bearing on the yield of a mine as a mine, which is the object of our consideration. A great deal more, however, depends upon the amount of working capital that is provided for a particular mine. An engineer may estimate that a certain sum of money will be required to do certain work, and will be probably within 25 per cent. of his estimate, but he cannot be absolutely certain that the results he hopes for will be realised—faults may be encountered, reefs may pinch out or become poor, and further developments may be necessary than have been provided for. In cases of this sort if a company be provided with a working capital equal only to what the engineer estimates as the cost of the necessary work, there is fair reason to suppose that this company will be a failure. As a matter of fact, in order to assure success, a working capital should be provided for every mine equal to at least twice the amount that will be required to equip the mine and commence treatment of the ore. I know of only one instance in which such a course has been pursued, and I consider it to be the best proof of my contention that can be found. In the case I allude to, the engineer estimated that £25,000 would be required. The promoters of the Company said we will make this £45,000, to be on the safe side. Over £25,000 was remitted to the mine, and at the end of the time the returns were not so good as had been anticipated. There was still money available, and a policy was pursued of careful and energetic management, and instead of the mine being shut down, as would undoubtedly have been the case if the capital had been limited, it was worked, and a sum of £18,000, roughly, was remitted to London from the mine during the ensuing twelve months.

To come now to the question of analysis of costs: it has long been the practice in collieries to analyse very carefully the costs of working, and on the fortnightly pay-sheets the cost of every branch of work (*e.g.*, hewing, tramming, winding, pumping, surface charges,

&c.) is worked out to the decimal of one penny per ton on the output for the period. By adopting this method, it is easy to see at a glance if the expenses in any department are being allowed to creep up, and the management is afforded an opportunity, directly any extra expense is incurred, to investigate the causes, and, when advisable, curtail the expenses.

This system, or one approximating to it, has also been adopted to some extent in the case of low-grade metalliferous mines in various parts of the world, with the result that working expenses have been very materially reduced, and ores, which under a haphazard system would not prove remunerative, have by careful management and a constant attention to details, been brought to a dividend-paying condition. There are numbers of mines, however, where these details are not attended to, and not a few where a manager is appointed and told at the same time that he will have nothing to do with the accounts beyond making out his pay-rolls and signing requisitions for such stores, &c., as he may require. This policy, when adopted, is suicidal to the interests of the mine, and although those few mines that will stand mismanagement may also survive neglect in this direction, the dividends that accrue to the shareholders will necessarily be less than they would be if economy were exercised throughout.

I heard of an instance lately which seems to illustrate the position very fairly. A man thought he saw a way in which he would be able to save a very prosperous mine some £7,000 a year, and he called upon a gentleman who was largely interested in this mine, and made his proposals. The answer was: "What does £7,000 a year matter to us?" In my opinion an urgent need exists for this analysis of expenditure in every mine, whether large or small, rich or poor.

It is also worthy of mention that a multiplicity of offices, and unnecessary expenditure incurred both abroad and at home, are often responsible for the non-payment of dividends, and numerous instances might be cited of companies which, under more economical administration, would have developed into dividend-paying concerns, that through lack of economy have had to suspend operations, and then reconstruct or liquidate; and this through no fault of the mine itself.

In a district like the Rand, where the mines are all under somewhat similar conditions, these details may be arrived at with perhaps greater accuracy than elsewhere, and it can be very clearly demonstrated that ore of a certain grade will pay under existing conditions, whereas a reduction in the cost of treatment of 1*s.* or 2*s.*

per ton will render large blocks of ground remunerative which previously would have been too poor to work. In a case like that, the greatest assistance to the district can be gained by publication of analysed working costs. There are very few fields, however, in which the conditions are so similar in the different mines as is the case in the Rand, and wherever lodes are mined, the costs will vary in different properties. However, even when this is so, great profit would result to the district if this analysis of costs were pursued, and the information were available to all interested in it and capable of availing themselves of the facts so obtained.

There is of course a distinction to be drawn between those mines which are developed—that, in fact, are making regular sales of ore or metal, either hand-picked or treated on the mine by battery, concentration plant, or furnace—and those which are in a prospecting stage and are only engaged upon development work. In either case, however, records can be kept and the costs of sinking, driving, &c., known.

In view of the large sums of money which are frequently expended on this prospecting work for very inadequate results as regards the distance sunk or driven, it seems reasonable to suppose that a very material saving in the expense of prospecting could be made if the information were available to those in charge as to the cost of the same work in other mines.

I shall perhaps be pardoned if I detain you for a few minutes on this question of mine accounts, a question which I should be very glad to see taken by some member as the subject for a paper, supplemental to the able one read before this Institution in 1897 by Mr. A. G. Charleton, as any steps which can be made towards unification of the system are important. I do not want you to understand that in my opinion the details which should be absolutely required of every mine need be very numerous, although on large mines minute details are of the greatest importance. I remember hearing a remark made by a Minister of the Crown in one of the Colonies when it was proposed at the instance of the Chief Surveyor to largely increase the survey vote. The Minister said: "Why the fellow thinks the Almighty only made the land to be surveyed;" and applying this remark to the case in point, it would appear that in no case should the accounts be of so detailed a character as to necessitate the employment of a number of clerks out of proportion to the value of a mine, for it is quite possible to have too much of a good thing.

In the first stage of a mine's career a minute and explanatory account of all expenditure can be kept in very simple form, from

which an elaborate set of books can be written up at any future time.

In every mine, however, accounts should be kept of each of the following:—Shaft sinking, cross-cutting, driving, rising, winze sinking, stoping, tramming, winding, crushing, amalgamating, concentrating, cyaniding or chlorinating, roasting, smelting, refining, management, office, London expenses, and all costs of labour and supplies should be debited to them. Repairs of machinery should also be charged direct to the working costs of the different branches, and in addition at least 10 per cent. should be written off the value of all plant year by year and carried to a reserve fund.

The question of amortization of capital is not so easy to deal with, but in view of the fact that the shares in mines are constantly changing hands, I do not think that any provision should be made in this direction, but rather that the shares should be looked upon simply as so much paper entitling the shareholders to what dividends can be made, and not as an ultimate asset when the mine is worked out. I know that this point is open to discussion, but I am inclined to lean to the view I have expressed.

If, however, this view be adopted, there is another point that requires very careful consideration, viz., how development work should be charged. It is not infrequently the case that the early developments on a mine are charged direct to capital account, a practice that is as manifestly wrong as it would be to charge it direct to profit and loss account. In view of the changes which are sure to take place in the value of an ore from time to time, it is necessary to keep a floating capital which will be available for developments during those periods when the mine is not doing well, and this can never be done if the working capital be permanently absorbed. In opening a mine I think that all development expenses should be carried to a suspense account called development account, and as soon as crushing or other treatment is commenced, a careful estimate of the ore developed should be made and the amount per ton calculated which this development has cost, exclusive of the main shaft and cross-cuts, and as the ore is treated or sold the development account should be credited with this amount per ton on the ore which has been mined. An amount which must be settled by the conditions which prevail—the depth of sinking and the probable life of the mine, should also be written off annually from the costs of the main shaft and cross-cuts.

The initial expenses on a mine are nearly always greater per ton of ore developed than they are later on, and consequently if this policy be pursued there is generally good reason to suppose that as

the mine continues to be worked the development charges will be reduced rather than increased, and therefore if this system be rigidly adhered to, no over-estimate is likely to be formed regarding the value of a property after the first six months' crushing or other treatment.

This charge per ton will necessarily vary in different districts and under different conditions, and may in certain cases be less than 1*s.* per ton, while in others—when the reefs are small and the ground hard, when the costs of supply are abnormal and the labour expensive and not of the best quality—the figure may be very large indeed.

I understand that on the Rand these charges amount to 3*s.* or 4*s.* per ton, and that in Rhodesia a sum of 3*s.* per ton is being charged against the ore in reckoning the total working costs at 18*s.* to 27*s.* per ton. A few other cases might be cited where the costs are somewhat similar, and then there are a vast number of mines which, to the public at any rate, afford no information on this subject.

It might appear at first sight that no difficulty would exist in obtaining this information from the accounts and balance sheets that are published year by year by the different companies, but unfortunately owing to the want of uniformity in the systems of keeping accounts it is very difficult indeed.

Even however on the basis of the profit and loss accounts published, and without criticising in any way the smallness of the sums written off for depreciation and development, there are many mines in which the cost of mining and treatment per ton is considerably higher than it should be.

It is a remarkable fact, and one to which I have already made allusion, that the working costs are generally greater on rich mines than on those which are poor, probably due in the majority of cases to a certain feeling of extravagance which is engendered by a sudden acquisition of wealth, but although it is doubtless admissible that the employés on a mine should participate to some extent in exceptional profits, it is no less true that unless all departments are carefully supervised, expenses are soon apt to creep up and thousands of pounds are lost which might have been saved.

I do not wish in my present capacity to criticise in detail the operations of different mines, but I do think that a comparative table showing the amount of gold won and the dividends earned—taken only from published returns—will be a fair illustration of the variations which exist in working costs, and I have been careful to take these only from mines which have been at work for some time and so have settled down into their regular stride:—

Mine.	Capital.	Per ton.	Tons crushed.	Gold recovered.	Dividends.
	£	oz. dwt. gr.		oz.	£
Lady Shenton	160,000	2 16 0	7,328	20,522	32,000
Great Boulder Prop...	160,000	2 1 2	41,043	84,350	200,000
Great Boulder Main Reef	120,000	1 14 15	8,222	14,248	12,000
Lake View Consols...	250,000	1 10 11	67,261	102,667	187,500
Champion Reef of India.....	220,000	1 10 8	89,271	135,412	242,000
Day Dawn Block and Wyndham	498,400	1 5 12	34,140	43,572	55,125
Mount Morgan	1,000,000	1 2 10	153,297	172,039	325,000
Brilliant G. M. Co. ..	520,000	1 1 23	34,031	37,381	84,500
Nundydroog	220,000	1 1 21	37,930	41,534	63,731
Ferreira	90,000	1 1 1	131,708	138,654	405,000
Waihi G. M. Co.	320,000	£3 5*s.*	77,909	£252,309	128,000
		oz. dwt. gr.		oz.	
Ooregum.....	265,000	0 15 11	67,942	52,585	51,350
Durban Roodepoort G. M. Co.....	125,000	0 12 6	122,750	75,924	100,000
Crown Reef G. M. Co.	120,000	0 12 21	185,179	119,245	288,000
Jubilee	50,000	0 9 20	63,419	31,269	50,000
Crown Deep.....	300,000	0 9 19	268,203	131,483	150,000
Jumpers G. M.	100,000	0 9 16	139,940	67,808	115,000
Geldenhuis Deep	300,000	0 9 13	284,700	136,108	225,000
New Primrose	300,000	0 8 23	265,551	119,350	165,000
Alaska Treadwell	1,000,000	$2 31c.	254,329	$586,857	60,000

We find instances recorded of the cheap working of certain colonial mines; for instance in the *Standard* of 2nd February, 1899, I notice our friend Mr. Lidgey mentions that amongst Victorian mines the Central Ellesmere crushed for the half year 6,900 tons for a yield of 911 oz. 1 dwt., an average of 2 dwt. 15 gr. per ton, which gave a profit on actual working expenses of £1,030 17*s.* 1*d.* The Thames United, though their stone averaged under 2 dwt. per ton from 4,147 tons, came out with a profit of £510. The Stewart Extended, which has to cart and pay for crushing, had an average of 2 dwt. 7·2 gr. per ton, and on a yield of 219 oz. 19 dwt. 11 gr. from 2,778 tons made a profit of £34 3*s.* 7*d.*

We cannot hope to make profits for London companies on returns such as these unless the mines are worked on a much larger scale than those quoted above, for the very obvious reason that the exigencies of the law demand that there should be certain officers of companies in London, and that these officers have to be paid. Then, moreover, while a mine under local conditions can be managed by a man at perhaps £5 a week, he is under the direct supervision of directors who are themselves mining men,—on the other hand a London

company has to employ a responsible manager who will necessarily have to be paid upon a more liberal scale. It thus becomes evident that to render low-grade mines payable under these conditions, the output must be large and the working capital in proportion.

Not less important than the question of accounts showing the cost of every operation on the mine is a careful record of the original value of the ore raised, the percentage of extraction by every process and the final quality of the tailings. The assayer and sampler should be the busiest men on a mine and a thorough knowledge should be acquired, not only of the cost, but of the results obtained at each stage of the operations.

In a gold mine we should know the value of the ore treated, the percentage saved by amalgamation, by cyaniding, by concentration, and by slime treatment, and the cost of each operation. We should also make ourselves acquainted with the losses which occur at each step in the process of extraction and compare these with the final difference between the assay value of the ore sent to the mill and the actual extraction.

In this connection I would comment on a system which is adopted on many of the most important mines of giving the returns. In the majority of mines the returns are published as so many tons crushed yielding so many ounces of gold, and so many tons cyanided yielding so many ounces. Of course the tons crushed and cyanided are not the same, and for purposes of comparison I would suggest that it would be much better to aim at giving, as is done in many of the Rand mines, the number of tons crushed, cyanided, and otherwise treated, with the gold won in each case—calculating eventually the average yield and value on the basis of the mill output.

There is necessarily a certain proportion of slimes which is permanently lost, and this in addition to the assay value of the tailings must be computed in arriving at the percentage extraction, which should be calculated in every case upon the value of the ore sent to the mill.

In a copper or lead mine the same patient and careful investigation is necessary, and the original sampling of the ore must be compared with the extraction. When there is a loss, it becomes necessary to trace the point or points in the process where it occurs, and the percentage and value of hand-picked, jigged, and finely concentrated ores should be determined as well as any loss which takes place at each operation. It is only by knowing the facts accurately that improvements can be made, and no mine has ever, I venture to suggest, reached that stage at which it is no longer

necessary to experiment with the view of either reducing costs or increasing the percentage of extraction.

In almost every mine we know, and speaking collectively of the Members of the Institution, I suppose there are few mines in the world with which we are not acquainted, we can recall instances where experiments have resulted in improved extraction and decreased cost, or we can see the direction in which experiments should be made in order to effect improvements.

Economy in work should be the order of the day, machinery should replace hand labour wherever it is possible, more especially in those districts where the cost of labour is high—piece work should replace day work wherever it is practicable; and in nearly every case it should be the aim of a manager to develop and treat large quantities of comparatively low-grade ore in preference to selecting small quantities of ore of high tenor.

In a word—if we wish the best possible results to be obtained, we must practise economy, a constant supervision of costs in every department, and an everlasting investigation of weak points in our methods of treatment; we must adopt improvements so soon as we are convinced that the need for them exists; and above all things, when we find that any process is a failure, we must discard it even if it is difficult to admit that we have been wrong.

Mr. Walter McDermott proposed a hearty vote of thanks to Mr. Cox for his address. He was sure the members would agree with him in saying that Mr. Cox's modest introduction was quite unnecessary, as was proved by the number of valuable suggestions embodied in the address, and which in themselves were quite sufficient to foreshadow the services they might expect from Mr. Cox during his year of office. He concluded by thanking Mr. Cox for his remarks as to his (the speaker's) services in the Chair, which were, however, in his opinion, of far too flattering a character.

Mr. J. H. Collins said it gave him pleasure to second the vote of thanks to Mr. Cox, as he felt quite sure that they would not be disappointed in that gentleman's record during his term of office.

The vote of thanks was carried by acclamation.

The President thanked the members for the manner in which they had received his address, and assured them that he would do everything he could to conduct the business of the Institution in a satisfactory manner while he occupied the Chair.

"Notes on the alleged Shortage in Cyanide Bullion."

By C. W. MERRILL, Assoc. Inst. M.M.

THE impression that the bullion returns from cyanide plants are, in spite of the observance of all possible precautions, uniformly less than the correct tonnage multiplied by the difference between the values per ton indicated by assays of the "charges" and "residues," seems to have become so general that it was thought that the accompanying plot, with a brief explanation as to how the figures therein were computed, would be of interest.

These figures are the results from the operations, from July, 1897, to November, 1898, of the tailings plant of the Montana Mining Company, Limited,* the capacity of which is 12,000 tons per month, which was installed and operated under the writer's supervision, and which has been working during all but the severest weather between the dates named, it being impossible to economically handle the store tailings in mid-winter,† the thermometer sometimes dropping as low as 50° below zero in this northern latitude, and the tailings freezing as hard as sandstone to a depth of several feet from the surface in consequence.

ESTIMATION OF OUTPUT BY DAILY ASSAYS OF CHARGE AND RESIDUE SAMPLES.

About 135 truck or car loads of tailings are hauled to the bins daily, each car containing approximately 3 tons of sand. From six to ten of these are weighed, and large samples are taken from each of the cars so weighed; the accumulation of these car moisture samples gives the daily average moisture sample. From the number of cars dumped, the average weight per load, and the moisture per-

* A description of the plant, which has since been enlarged, appears in the *Journal of the Association of Engineering Societies* (U. S.), May, 1898, vol. xx, pp. 324—326; also in the *Engineering and Mining Journal* of April 16, 1898, and *Mining and Scientific Press*, April 30, 1898.

† The plant was however operated for a short period, during November, 1898, when the thermometer ranged from zero to 15° below.

centage determined from the average sample is figured the number of dry tons treated daily.

The sand passes from the cars to the bins and is withdrawn by a 24-in. belt conveyor, one of which discharges into a rotary distributor over the centre of each sand vat. The stream of sand as it leaves the belt is sampled three times per hour during nine hours, which period is the average time consumed in filling a vat with 400 tons of tailings. The large "charge" assay sample so obtained is quartered down and 4 assay tons of the dried sample are taken for duplicate melts and weighings of gold and silver, care being taken that the lead buttons from each melt of 2 assay tons shall weigh approximately 30 grams. If the variation in the weights of gold obtained from these two melts of 2 assay tons each does not exceed $\frac{1}{50}$ milligram (equivalent to 20 cents per ton), the average is taken as representing the value per ton of the tailings charged that day; otherwise the assays are repeated in duplicate until such a concordance of weights is obtained.

After treatment the residue in the vat is sampled by removing 16 cores from top to bottom by means of the sampler (Fig. 1). The sample thus obtained, weighing several hundred pounds, is in turn cored with a small tryer, yielding about 20 lb. of residue, which is dried, mixed, and quartered for the final assay sample. Of this, 8 assay tons are taken for assay, the doré buttons from each melt of 2 assay tons being weighed separately and parted in pairs so as to give duplicate gold weighings from 4 assay tons each. If these weighings differ by more than $\frac{1}{50}$ milligram (equivalent to 10 cents per ton) the assays are repeated. All gold weighings are made on an Oertling 12A balance, sensitised to $\frac{1}{100}$ milligram.

The monthly outputs calculated on the basis of the preceding determinations are shown in the upper line of the accompanying plot, the gold being plotted to the right of the datum line and the silver to be left.

Estimation by Daily Solution Assays.

The total amount of unprecipitated or pregnant solution flowing from the sand vats daily is measured, sampled and assayed, as is also the corresponding precipitated or barren solution.

Of the unprecipitated solution from 4 to 8 assay tons are measured and evaporated to small bulk in ordinary saucers standing on a hot plate. Each saucer holds 5 or 6 assay tons, if larger volumes require to be taken a corresponding number of saucers are used and the contents, when sufficiently reduced, are poured into one. A mixture of litharge, silica, and charcoal is then added and

the whole heated till dry, when the contents of the saucers, after cooling, are mixed with sodium carbonate and charged into a crucible. Any residue adhering to the saucers is moistened with dilute nitric acid and taken up by a small piece of filter paper which is then sprinkled with litharge and added to the crucible. The mixture is then melted, the resulting lead button cupelled, and gold and silver determined. Duplicate solution assays are not usually made, but when so made the results almost invariably agree very closely. The sodium carbonate is not added to the solution with the rest of the flux, because it would cake on the inner surface of the dish and make the mixture difficult to remove. Of the precipitated solution, from 8 to 24 assay tons are similarly prepared for the furnace, as many as 40 assay tons being occasionally taken.

These solution assays are of great practical utility, not only as guides to the efficiency of the precipitation, but also as giving warning of the presence of reducing material in any charge several days before the residue from such a charge is available for assay, and they are made primarily for these reasons.

The estimation of the values recovered monthly on the basis of these solution assays is shown on the lower line of the plot.

Monthly Bullion Assays.

The monthly bullion assays are made with the usual precautions, including determination of the cupel absorption, check for volatilization of silver, &c.; the results are shown in the centre line of the plot. The shortage of bullion in the first clean-up is usual in cyanide plants, the deficiency being made up in the succeeding months.

Comparison.

The output of gold determined by daily assays of charge and residue is seen to be $276,499·70, while the gold in the bullion obtained was $284,856·35 or 3·02 per cent. greater. The silver called for by daily assays was $51,596·51, while that obtained in the bullion was $52,572·27 or 1·85 per cent. more. Comparing the total values we find $328,096·21 called for by daily assays, and $337,428·62 obtained in bullion, or an excess of 2·84 per cent. over the estimate, a highly satisfactory concordance.

The estimates of precipitation have also averaged somewhat less than the gold and silver obtained, the value of the gold being slightly greater than that called for by daily assays of charge and residue, and that of silver materially less. The deficiency in silver

is partly due to the fact that the lead buttons from the solution assays, which are made primarily as guides to the efficiency of solution and precipitation, are placed in the rear row of the muffle during cupellation.

The bases of values, $20·30 per fine oz. of gold and $0·50 per fine oz. of silver, were taken for reasons of convenience in computation, their values not affecting the comparison.

The ton referred to is the United States ton of 2,000 lb., and the assay ton contains 29·166 grams, or the same number of milligrams as the ton contains of troy ounces.

DISCUSSION.

The President thought that the paper was a very useful one, and the author appeared to have proved, so far as the Montana Mine was concerned, that there was no shortage in the bullion there. He would have liked to have seen the monthly returns given in detail, and perhaps it was not too late to have them embodied in the paper. The author's remarks about the Montana did not, of course, necessarily apply to all other mines, and it would be interesting to hear the opinions of some of the cyanide experts present with regard to the work which had been accomplished in other places.

Mr. Butters said that he did not presume that Mr. Merrill wished them to suppose that he obtained 2·84 per cent. more gold and silver than was contained in the ore, and it must be assumed that there was something wrong in the assays or the calculations. As a cyanide man he should say there was more likely to be a shortage of 2·84 per cent. than an excess of that amount. There were bound to be losses in such work, and no assayer could show an excess of gold, for such a thing did not exist. He understood that the zinc process was used at the Montana, and they were all aware that it was impossible to clean up without some loss of gold when that process was in operation; about 1½ per cent. was the usual loss. In slimes' treatment he had seen a loss of 30 per cent. month after month. The losses in question were not unaccountable losses, but occurred in the mechanical handling; it was certain that they could not handle gold without losing some of it; the amount depended upon the skill of those performing the work. Some of the losses were got back when the works were over-hauled; gold would be found everywhere from the sumps where precipitation took place

to the chimney stack of the furnaces where the gold was melted, and even in the sweepings of the yard of the melting house. It was lack of knowledge in the early days that caused the impression that the losses which occurred were unaccountable. He did not want any member of the Institution to plume himself on having an excess of gold; he might take credit to himself for having got nearly all the gold out, but he certainly could not get out more than was in the ore; and we must assume that the author's actual extraction was below what his correct theoretical extraction should have called for.

Mr. Franklin White considered the paper was a very useful one, because there was an impression that the actual returns from cyanide work did not, and could not be made to agree with the returns that might be obtained theoretically. He agreed with Mr. Butters, that they could not accept the author's statement with regard to an excess of bullion as satisfactory, and it was just as much an error as if it were a deficit. He should like to know something more about the value of the ore that was dealt with; he inferred it was about 5½ dwt., and the excess of gold in that case would represent an error in the assay of about 4 gr. per ton. In looking through the paper to see where the error might have crept in he noticed several points; the first was that of the moisture which was referred to by Mr. Butters. It also appeared to him that the system of sampling the residues was liable to give too low a value, and a fair proportion would not be obtained from the bottom of the vat, which was naturally the richest. It would be interesting to know whether the monthly returns showed a general excess of gold or whether it was higher in some cases and lower in others. If the general tendency was to err on the side of excess, it would show that there was a standing error in the calculations somewhere, either in the tonnage or the system of assaying. It was evident that instead of an excess there should be a deficit, for, as Mr. Butters had remarked, there was always a loss in mechanical handling. He would have preferred to have seen an acknowledgment of a deficit than the claim for an excess which had been put forward by the author. On the Rand they were coming to the conclusion that the general source of error lay in inaccuracy in estimating the tonnage, and not in the assaying.

Mr. McDermott thought that the criticisms which had been passed on the paper would meet with general approval, seeing that it was impossible, if the average value of the ore treated was correctly estimated, to have extracted more gold than it actually contained. He thought the title of the paper was hardly justifiable;

"alleged shortage" was quite an incorrect expression, as the general experience was that there must necessarily be a shortage of bullion, either to a large or small extent. The paper was, however, interesting as showing how on a certain system of sampling and estimating of weights an apparent gain was made, whereas in other districts, on a different system, a larger loss than actually occurred was sometimes shown. The author stated that the stream of sand as it left the belt was sampled three times per hour during nine hours; it might be easy to get slight variations, although it was difficult to see how it was that the ore should be continually under estimated, as seemed to be suggested. He thought that probably the solution lay, as suggested by Mr. Franklin White, in the method of sampling; in taking a sample through a depth of ore which had been packed down it was difficult by the use of an auger to get a real proportionate sample, and it could not be depended upon to give uniform results. It was possible that Mr. Merrill's paper had conveyed an impression different to that which the author intended, but he could not help agreeing with Mr. Butters and Mr. Franklin White that they could not accept his conclusions as being correct.

The President said that the statement made by Mr. Butters and other speakers that it was impossible to get out more gold than was in the ore was manifestly correct, and he should imagine that the method of sampling was responsible for the error. No doubt Mr. Merrill would further investigate the subject, and lay his views before them when he replied to the points raised in the discussion.

"On the Chemical Reactions involved in the Amalgamation of Silver Ores."

By HENRY F. COLLINS, M.Inst.M.M.

THE experiments of Laur, Bowring, Uslar, Huntington, and others have proved that the principal reagents in the patio process are mercury and cuprous chloride. The former combines direct with such portion of the silver contents of the ore as may exist in the metallic condition or combined with haloid elements, while the latter (formed *in situ* by the action of mercury on cupric chloride) plays the principal part in decomposing silver sulphide.

Most metallurgical writers appear to have taken it for granted that as regards the chlorides of copper, at all events, the reactions which take place in the pan are similar to those of the patio process, and the conclusions of Hague, based upon his own experiments, have been accepted without question. These conclusions are as follows:—*

1. That NaCl and $CuSO_4$ give rise by double decomposition to the formation of $CuCl_2$.
2. That metallic iron reduces $CuCl_2$ to Cu_2Cl_2.
3. That both $CuCl_2$ and Cu_2Cl_2 assist in chloridizing Ag_2S, as well as in decomposing galena and blende.
4. That $CuSO_4$ forms a "small quantity" of copper amalgam.
5. That the quantities of NaCl and $CuSO_4$ are in any case too small to have any effect.
6. That mercury and iron, aided by heat and friction, are the chief agents.

The general opinion among metallurgical writers as to the reactions which take place in the pan follows Hague, and is expressed by Schnabel† as follows:—

"The essence of the process consists in the fact that quicksilver in presence of iron liberates silver from silver sulphide, that silver chloride is decomposed by iron and quicksilver with separation of metallic silver, and that from the mercurous chloride thus formed the quicksilver is again set free by

* *Mon. U.S. Geol. Survey, 40th parallel*, vol. vii, pp. 275—293.

† *Handbuch der Metallhüttenkunde*, 1894, vol. i, p. 665.

means of iron. The metallic silver liberated in all these ways, as well as that existing in the ores in a native condition, is taken up by quicksilver.

"From the reagents (copper sulphate and salt), added in small quantities, copper chloride is formed, which salt is first reduced by iron to cuprous chloride and then liberates silver, not only from silver sulphide, but also from any complex arsenical and antimonial sulphur compounds present; as already described in connection with the kröhnke and patio processes."

The functions of the iron of the pan itself, together with that existing in the pulp in a finely divided condition to the amount of 3 to 10 lb. per ton of ore and resulting from the abrasion of battery and pan castings, are summed up by both Hague and Egleston as follows:—(1) Direct reduction of AgCl to metal; (2) reduction of calomel; (3) direct reduction of Ag_2S; and (4) reduction of $CuCl_2$ to Cu_2Cl_2; and both concur in Hague's conclusion "that mercury and iron aided by heat and friction are the principal reagents in the extraction of the precious metals in the Washoe process."

No exception can be taken to this conclusion as regards docile ores, which require neither salt nor bluestone for their reduction. On ores which are at all base, however, extraction is much improved by the use of these substances, and their use forms an essential feature of the pan amalgamation process as ordinarily conducted. In the author's opinion the part played by bluestone in the pan has been very generally misunderstood; the formation of Cu_2Cl_2 can only be accidental and momentary, and all the copper added is reduced to metal almost as soon as it is dissolved.

It is indeed well known that practically all the copper added to a pan charge is eventually taken up by the quicksilver and enters the amalgam, and further that this precipitation of metallic copper in some way aids the reactions. Thus Eissler states:* "the lower the fineness of the bullion produced the higher was the percentage of silver extracted from the ore," and Hodges in his paper on refining copper bullion† lays stress on the fact that all the copper added to the pan is eventually found in the bullion. It does not seem to have occurred to anyone, however, to trace the stage at which the copper becomes amalgamated, or to doubt that Cu_2Cl_2 is formed by the action of even a large excess of iron upon $CuCl_2$. The term "large excess of iron" is used advisedly, because as a rule the iron contained in the pulp in a finely divided condition when it enters the pan is more than sufficient to precipitate the whole of the added copper without allowing for the removal of any iron from the pan itself by abrasion or corrosion.

The classical experiment of Hague, upon which so much miscon-

* *Metallurgy of Silver*, 1891, p. 157.

† *Trans. A.I.M.E.*, vol. xiv, p. 731.

ception of the action of copper sulphate has been based was as follows :—

"Two grams of powdered blende, 1 gram iron filings, 5 grams NaCl, 7 grams $CuSO_4 5Ag$ were placed in a flask with a little water and kept at 90° C. for a few days. The iron filings rapidly disappeared, metallic copper was precipitated but redissolved probably by the chloride of copper present, the subchloride being produced. Later the iron was thrown down as a basic salt;* the oxide of zinc estimated in the solution was 0·3250 gram."

Leaving out of consideration the action of $CuCl_2$ upon ZnS, it would appear that Hague and those who followed him drew from this experiment the inference that Cu_2Cl_2 is formed during amalgamation. Under the conditions of Hague's experiment, however, three equivalents of copper are present for every two equivalents of iron, and therefore there is not sufficient iron present to precipitate all the copper as metal which would require equal equivalents. Moreover, the presence of blende, itself inert, prevents the experimenter from seeing what takes place. In repeating this and all other experiments having any bearing on pan amalgamation, the great preponderance of iron over all the other reagents under actual working conditions should be clearly kept in mind.

In investigating the subject the following experiments were made by the author.

Experiment 1.—Repeated Hague's experiment in triplicate, leaving out the blende, and instead of standing for some days, boiled up for periods of 15 minutes, 30 minutes and two hours respectively; then determined the undissolved copper. The amounts of precipitated copper remaining undissolved were respectively 55 per cent., 24 per cent., and 12½ per cent. of the total copper present, showing that the resolution of precipitated copper in large excess of $CuCl_2$ is a very slow process. In the pan of course there can be no such thing as excess of $CuCl_2$.

Experiment 2.—In presence of free metallic iron in excess, Cu_2Cl_2 dissolved in excess of NaCl was rapidly reduced to the metallic state, quite as rapidly in fact as a solution of $CuCl_2$ or $CuSO_4$, containing the same amount of copper.

It appears, therefore, that Cu_2Cl_2 in the pan charge, whether formed accidentally or introduced of set purpose, would be almost instantaneously reduced by the large excess of iron present.

Experiments 3 *and* 4.—A measured quantity of a copper-salt solution was ground with sand in an iron mortar, for periods of five and ten minutes respectively. After five minutes' grinding, only 6 per

* By absorption of atmospheric oxygen.—AUTHOR.

cent. of the total copper was found to be still soluble in water, and after 10 minutes' grinding only 2 per cent., the whole of the remainder having been precipitated.

The supposed existence of cuprous chloride in pan pulp is, therefore, a myth, and the active reagent introduced by the addition of bluestone to the pan charge is finely divided *metallic copper*, which soon becomes amalgamated by minute globules of mercury, without, however, losing its power of acting on silver compounds.

A series of experiments was next undertaken with a view to ascertain the relative activities of iron, copper, mercury, and cuprous chloride in liberating silver from its compounds. The compounds investigated were the chloride and sulphide. The chloride was freshly precipitated *in situ* from a weighed quantity of $AgNO_3$, so as to get it as finely divided as possible, while with the same object in view the silver sulphide was prepared in the wet way, and ground with weighed excess of clean quartz sand to an impalpable powder, the proportion of silver in which was accurately known.

In order that the experiments might be under exactly similar conditions, a shaking frame was prepared to hold six closely stoppered bottles, into which the silver compounds and reagents were placed, and when the effect of heat was desired the whole frame, with its contents, was plunged into a can of water, and boiled for the desired period.

Experiments on Silver Chloride.

Experiment 5.—To equal quantities of freshly precipitated AgCl with excess of salt were added equal equivalents of the following substances, and after 10 minutes' agitation the excess of AgCl was dissolved in ammonia, and the precipitated silver determined. The results were as follows:—

Reagent added.	Per cent. of total silver precipitated.
1 equivalent of mercury	12·56
1 „ „ iron filings	28·80
1 „ „ copper precipitate	88·50
1 each copper sulphate and iron filings	97·10

As the much more rapid effect of the precipitated copper was judged to be due to its more finely divided state, another experiment (No. 6) was made by which to equal weights of freshly precipitated AgCl in excess of salt equal surfaces of the three metals were exposed for equal times at a boiling temperature. The results were as follows:—

Metallic surface.	Per cent. of total silver precipitated.
Mercury	13·2
Sheet iron	91·9
Sheet copper	93·3

It appears, therefore, that for equal surfaces exposed metallic iron and copper are about equal in their decomposing effect on silver chloride, while mercury is only about one-seventh as powerful.

These experiments throw light on the well-known fact that docile chloride ores can be treated quicker with a better extraction and less loss of mercury in the iron pan than in the patio. The results on sulphide ores, however, are known to be quite different, and the next series of experiments on an artificial mixture of sand with silver sulphide was directed to investigate this difference. The difficulty of determining the respective amounts of metallic silver and of unaltered silver sulphide in the residues was got over by taking up the former with excess of mercury, panning off, cleaning, and retorting the globules of mercury in a small retort 1 in. in diameter, specially made out of a piece of cast steel, and provided with a screw stopper and a long ¼ in. iron pipe. Metallic lead was added to the mercury before retorting, so as to give a tangible residue, which was then wrapped in lead foil and cupelled.

Experiment 7.—Silver sulphide was ground with excess of sand, mercury, salt, and the following special reagents in a Wedgewood mortar, and the silver reduced to the metallic state determined as above. Eight sets of experiments were necessary in order to give anything like concordant results, the mean being as follows:—

Special reagent added.	Per cent. of silver reduced to metal.
Iron filings	2·60
Precipitated copper	5·99
Cuprous chloride	7·62
Cu_2Cl_2 and iron filings	10·66
$CuSO_4$ and iron filings	3·20
None	2·00

The above results not seeming to point to any definite conclusion beyond proving that cuprous chloride has a more powerful effect than metallic copper or iron, another set of experiments was made in bottles, contained in the shaking frame already referred to.

Experiment 8.—To equal quantities of finely divided silver sulphide were added excess of mercury and salt and the special reagents mentioned below. The total time in contact was 80 hours, of

which a total of 26 hours was at the boiling temperature with constant agitation. The silver reduced to metal was determined as before, and the results showed :—

Special reagent added.	Per cent. of total silver reduced to metal.
Iron filings	32·8
Copper precipitate (fresh)	57·0
Cu_2Cl_2 and copper filings	72·3

This experiment confirms the preceding, and shows clearly that the effect of metallic copper upon Ag_2S, though more powerful than that of iron, is less powerful than that of Cu_2Cl_2. This explains the rooted aversion which patio and fondo amalgamators have to the use of iron in grinding unless counterbalanced by a comparatively enormous subsequent excess of bluestone, and the undoubted fact that on raw sulphide ores the patio or the fondo will extract a better percentage than the pan in which the addition of bluestone in even large amount can only result in the formation of metallic copper, but as already shown never in that of the more active Cu_2Cl_2. The extraction of 72 per cent. of the total silver present as Ag_2S in only 26 hours' boiling by the aid of a small excess of Cu_2Cl_2 is a sufficiently remarkable result, and shows why such ores can be worked in the fondon with some approach to technical success, as in all probability something like half of the total extraction would have taken place in the first six or eight hours. On such an insoluble substance as Ag_2S the rapidity of extraction probably depends almost entirely upon the degree of subdivision of the particles.

Conclusions.—The conclusions which may be drawn from a consideration of the above experiments are the following :—

1. When treating "chloride" silver ores the iron pan will give at least as perfect an extraction as the patio with far less loss of mercury, owing to the fact that the affinity of iron for the chlorine or other haloid element, combined with the silver, is so much greater than that of mercury (Experiments 5 and 6).

2. When treating such ores in the pan, the reactions are not sufficiently accelerated by the use of bluestone to compensate for the increased cost. Similarly, the increase in rapidity of extraction brought about by the substitution of the copper fondo for the iron pan is not nearly sufficient to offset its increased cost (Experiment 6).

3. When treating sulphide ores, however, the extraction is much accelerated and increased by addition of bluestone (Experiments 7 and 8).

4. The bluestone when added to the pan charge is within a very few minutes reduced to metal, partly by the finely divided iron present in the pulp, partly by contact with the large masses of that metal in the form of pan castings, which become corroded as well as abraded. The copper so precipitated amalgamates with the mercury, but whether free or amalgamated, acts upon silver sulphide more energetically than either metallic iron or mercury, or both combined (Experiments 7 and 8). No cuprous chloride is formed in the pan, or if formed temporarily by secondary action of precipitated copper upon undecomposed cupric chloride, it is reduced to metal by the excess of iron present long before it can exercise any effect upon the silver sulphide present (Experiments 1, 2, 3, and 4).

5. That nevertheless cuprous chloride has a still more powerful effect upon Ag_2S than metallic copper, however freshly precipitated (Experiments 7 and 8).

6. That this powerful effect of Cu_2Cl_2 explains the observed fact that upon raw or badly roasted sulphide ores, the *fondo* (in which Cu_2Cl_2 is constantly regenerated for re-use) gives a better extraction than the *pan* (in which Cu_2Cl_2 if accidentally formed, is almost at once decomposed).

In conclusion the author wishes to acknowledge the valuable aid which he has received in the conduct of these experiments from his late assistant, Mr. Alexander W. Marshall, Assoc. Inst.M.M.

Author's Apology.

Anticipating criticism of these experiments as unpractical, on the ground that they refer to processes which are either obsolete or rapidly becoming so, the author thinks it well to recall the fact that over one-fourth of the world's total output of silver is still produced by amalgamation processes, which in countries where the ores and conditions are suitable as in parts of Mexico, Columbia, Bolivia, and Peru, are likely to hold their own as against smelting processes for at least another century to come.

DISCUSSION.

Professor Huntington said that it was 17 years since he made the experiments mentioned by the author, and they were consequently not very fresh in his memory. They were directed more particularly to the patio process, in which iron did not take part, and therefore did not bear very directly on the experiments put

forward by Mr. Collins. He recollected that one important point which impressed itself upon him was that one had to be very careful what reagent was chosen in making the investigation. Some of the results obtained by previous investigators were found to be quite erroneous, owing to the fact that the reagent used caused the chemical reactions to be reversed. He believed the reagent in question was ammonia, and it had the effect of reversing the reaction entirely, and also set up a secondary reaction, so that one would be entirely misled if ammonia was used as a reagent in carrying on the investigation as to what had taken place in the process. He mentioned the matter because it was necessary to be particularly careful in studying the effect of the reagents used in investigating processes. He thought Mr. Collins's paper was a very useful one, and cleared up several points in connection with the amalgamation process, which, as the author had stated, was a long way from being dead.

Dr. Simon said that he had made experiments which confirmed Mr. Collins's statements, and had found that cuprous sulphate precipitated copper almost instantaneously and formed a very nice copper amalgam. He thought that the action of cuprous sulphate on silver ores was not to be looked for in the action of the copper on the silver salts which were present, but in the powerful reducing action of the sulphate of iron which was in a nascent state as soon as the iron took the place of copper in the cuprous sulphate.

He agreed with Professor Huntington that the experiments were very useful, and should contribute to the clearing up of that process, which was rather a cooking process, in which little bits of this and little bits of that were added. It was only by scientific research that a proper method of working would be indicated, and they must all feel grateful to Mr. Collins for having made the experiments and for having communicated them to the Institution.

Mr. Butters thought that the paper was a very interesting one, and did not agree with the author in describing the process as obsolete. Although the pan process was not the "cure-all" it used to be, it was an extremely useful process, and was one that would always be used by metallurgists under certain circumstances. The author mentioned that the only possible competitor to the process in such countries as Mexico, Columbia, Bolivia, and Peru was smelting; he had made some experiments on a silver ore from Mexico, and had obtained some very interesting results. They were treating the ore in a pan, concentrating, roasting the concentrates, and afterwards cyaniding the tailings; he found that it was only necessary to cyanide the stuff direct to get just as good results as was

obtained by the combined process. Some of the silver extractions were very surprising; he made several hundred experiments, and was astonished at the results. He found that with some ores, heating the cyanide solutions was a great assistance and increased the extraction in every case, while in other cases heat was of no benefit. As Mr. Franklin White was aware, the heating of cyanide solutions had been under discussion in South Africa for some time past, but he believed it was the first time it had been used in the case where silver was one of the elements to be extracted. He had a small plant working the Siemens and Halske process at present in Mexico, where they were extracting silver and gold with heated solutions, and he thought there was a good deal to be learned with regard to silver extraction in connection with the cyanide process; he considered it was going to be a great competitor to the patio and the pan.

Mr. McDermott said that with regard to the author's footnote, they did not wish to discourage papers on processes because they were less generally used than was formerly the case. As to the future of the pan process, they would (notwithstanding Mr. Butters's remarks) keep it in their minds for different parts of the world until it was displaced. It seemed extraordinary to him that cyanide performed the work it did—it appeared to attack anything under any sort of conditions, and the new idea of boiling it apparently still further increased its efficiency; he would therefore hesitate to put any limit on its possible application in the future. The point mentioned by the author in regard to the reduction of the bluestone seemed borne out by experience in South America and in parts of the United States where copper, instead of iron, pans were used.

Mr. J. H. Collins thanked the members, on behalf of his son, for their reception of his paper. He thought the author's experiments justified the "ignorant Mexicans" in their opposition to some of the scientific people who had been urging them to use stamps for their silver ores. He remembered seeing two large batteries at work in the neighbourhood of Zacatecas, but they did not seem to do as well as the patio process. Although the Mexican could not give reasons for his belief, he knew that the scientific people were sometimes wrong, and therefore kept to his old method, notwithstanding the immensely greater cost of grinding his ore with the ancient arrastras. Copper pans were used, as the Mexicans were well aware that the silver could be got out better than with iron pans. In many cases they did all they could to keep iron from touching the patio piles; they even carried it to excess by making their mules tread the piles without shoes.

EIGHTH ORDINARY MEETING, 17th May, 1899.

Mr. S. HERBERT COX (*President*) in the Chair.

The Minutes of the previous meeting were read and confirmed.

"The Lucknow (N. S. W.) Gold Field."

By A. R. CANNING, Assoc.Inst.M.M.

INTRODUCTION.

FEW goldfields have a record like Lucknow. With a length of little more than half a mile, and a width of somewhat less, the area therein contained has produced not less than 15, and probably many more tons of gold. Whilst such an output necessarily deserves mention, the remarkable formation in which the gold is found and the unique conditions encountered in the working of the ore bodies offer the greater incentive to supply a full description of the field and justify the writing of this paper.

Locality.—Lucknow is situated 6 miles from Orange, a thriving little town on the Western Railway line of New South Wales, and distant some 200 miles from Sydney. The elevation above sea level is just under 3,000 ft.; the climate is excellent. The rainfall is generally good, only on rare occasions causing any anxiety as to the requisite quantity of water for battery and boiler purposes.

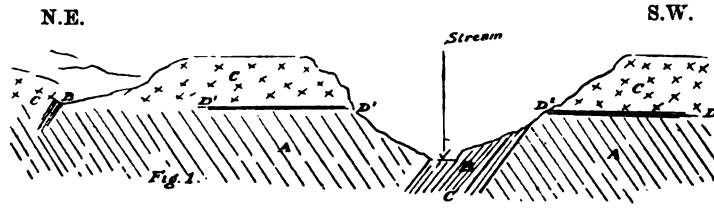
General Geology.—The predominant rock of the district is locally known under the general and convenient, though probably incorrect, term of "diorite." Through this there runs a dyke of serpentine, the strike, dip, and width of which varies considerably; the strike is for the most part N.W. to S.E., the dip for the first 500 ft. between 50° and 60° out of the horizontal; and the width some 300 ft., closing in towards the N.W. to only a few feet.

A second belt of serpentine is supposed to exist parallel, or nearly so, to the first band; but it is difficult to give this as an absolutely certain fact owing to the secrecy maintained with regard to the mine plans of an adjoining property. Half a mile to the N.E. a narrow and ill-defined seam of serpentine can be seen traversing a stream bed.

Whether further parallel dykes of this rock exist cannot be

proved, for the surface of the country gives no assistance to the searcher owing to an extensive capping of basalt.

This basalt, an important feature of the district, seems to have flowed from a group of hills lying 7 miles to the west, and which have been proved as former points of eruption. The highest peak is 4,565 ft. above sea level.



Section of Valley.—*A*, *A*, "diorite"; *B*, *B*, serpentine; *C*, *C*, *C*, basalt; D^1, old river bed, non-auriferous; D^2, old river bed, auriferous.

Auriferous Deposits.

The gold occurs under three different conditions, namely in :—

(*a*) Recent surface alluvial.
(*b*) Alluvial underlying basalt.
(*c*) In veins found in the S.W. body of "diorite."

(*a*) *Recent Surface Alluvial.*—This is the usual reef shedding, due to degradation of outcropping ore bodies. Whilst of little interest beyond the value won from this source, these deposits, nevertheless, offer further evidence of the immense richness of the veins from which they are shed. It is remarkable that, whilst during the period prior to the basalt flow the watercourses carried away large quantities of disintegrated reef to considerable distances, subsequent erosions do not seem to have removed the broken auriferous veinstone more than a few yards. The watercourse now flows N.W. instead of S. as formerly, that is to say, the old river ran across the auriferous zone, whilst the present stream is found parallel to the gold line and distant from one to two hundred yards. No payable wash is found down the existent valley. Again, whilst in most cases alluvial lying over the reef from which it is shed throws little light on the value of the stone constituting the underlying lode, in this case it would appear to be otherwise. Here it is plain (see Fig. 1) that recent erosion cannot have reduced the ore bodies more than from 10 to 20 ft., for the bed of the original stream did not lie at a greater height above than this, and it is natural to suppose that all gold shed at the time that the ancient river was running, was carried away by its waters and deposited as described below.

Owing to these features the surface alluvial found has been limited in area but immensely rich, and the history of the early days of Lucknow is sensational; over 10,000 men at one time got their living, and not a few their fortunes here some 30 years ago. To trace the source from which this gold was derived offered no difficulties, but followed rather as a natural sequence.

(*b*) *Older Alluvial.*—These beds were deposited by a stream which ran across the auriferous zone, denuding the outcropping reefs and carrying away the gold-bearing quartz. This stream bed was subsequently buried under the lavas that form so important a part of the district. A very large amount of gold has been won both on the Lucknow field and at Forest Reefs, 6 miles away, and again at a spot distant some 20 miles. The wash dirt varies from 6 in. to 3 ft., averaging 2 ft. in thickness, and ranges from 50 to 500 ft. in width. This has been mined by shafts through the basalt and by an adit. The wash lying immediately south of Lucknow is very little worn, being almost angular; there is no fine drift sand, and the mining is easy. The value of the gravel was seldom less than ¼ oz., and reached as high as 5 oz. bullion per load, and large quantities of gold have been extracted from it. This ground was abandoned 25 years ago, but is just now again receiving attention.

(*c*) *Reefs.*—In the main or lower body of diorite, there are found quartz reefs lying for the most part parallel with one another and differing but little except in size and strength. These reefs occur at irregular intervals, and, their general strike being east and west, they meet the contact joint between the diorite and the serpentine at an angle of about 60°. At or near this point of junction gold is found in conjunction with calcite, and it is consequently along the contact joint that the whole interest of the field is centred. So far as is known, the second body of diorite does not carry gold-bearing reefs, so that in the following description only the main or lower body and the serpentine immediately overlying it will receive mention, and the line of contact between these two rocks will be known as the "contact joint" or "joint."

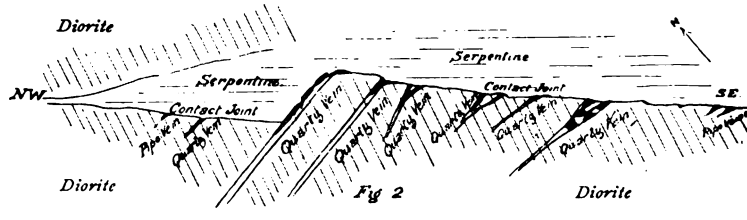


Fig 2

Sketch Plan, showing quartz and "pipe" veins; pay shoots marked black.

Along the outcrop of the "joint" after the removal of the recent alluvial some 20 apparently distinct lodes were discovered. Some of these were not more than 50 to 60 ft. apart, others lay several hundred feet away from the next. About 3,000 ft. divided the most south-easternly from the extreme north-westernly body. Most of these veins on reaching the contact joint overflowed along it in a south-easternly direction, and the gold for the most part existing in this overflow, the early miners on the field regarded the contact joint as the main channel and the auriferous body found along it was known as "the lode." In reports made by a Government geologist and others at that time allusion is made to this "lode," and as they form the only source from which to gather a reliable description of the exceedingly interesting appearance of the ground exposed by the working of that time (20 years ago), the writer may be pardoned for quoting a paragraph from Mr. Wilkinson's report to his Government; this report is dated May, 1881. "The reefs occur at the junction of serpentine and hornblendic felsite, the latter in places passing into diorite. Along this line of junction is what the miners term the 'lode,' which at the surface is a fissure 6 ft. or more in width, extending in a direction S. 50° E. for a distance of 50 chains. It is filled with a red sandy ferruginous clay containing hard silicious accretions of irregular shape, locally termed 'clinkers.' This lode dips to the north-east at an angle of about 65°, though in some places it is nearly vertical. The hornblendic felsite forms the foot wall and the serpentine the hanging wall. In the felsite at various distances along the lode are quartz veins from a few inches to 6 ft. thick coming in from the west and abutting against the lode which they appear to follow down, and form irregular quartz 'pipes' or 'shoots' dipping diagonally along the lode towards the east. These veins have only been found to contain gold where they occur in the lode, and form the 'shoots.'"

Below the influences of surface water and other oxidising agents, that is to say at a vertical depth of from 150 to 200 ft., this joint lode disappeared; but as one excellent example occurs in an unoxidised state between the 400 and 500 ft. levels, a clue is given to the curious formation described in the foregoing paragraph. At the level at which the sulphides began to appear some of the smaller ore bodies ceased to exist; but there can be little doubt that in their form and origin they were similar to those that reached greater depth. All these quartz reefs on nearing the contact joint either become changed into carbonate of lime or, in the case of the very strong reefs, carry calcite along the north side; this calcite is almost invariably auriferous and is often immensely rich in gold. The dis-

tances from the joint at which the gold-bearing stone exists varies greatly, ranging from 4 or 5 ft. up to 200 ft. Some of these reefs have only one shoot of gold-bearing calcite, others have two and three shoots, the reef intervening between them being a valueless quartz.

Whilst the contact joint dips north these quartz reefs are either vertical or dip slightly to the south, and where such is the case the gold deposit is not unfrequently found on the underside of them, eventually falling away into "droppers."

The strike of the veins varies somewhat, the more important bodies bearing east and west, and the smaller ranging between that and N. 25° E. and S. 25° W. Bifurcations occur in more than one ore body, the two splits in the vein being found to rejoin after 100 to 200 ft. Incidentally it may be noted that whilst these branches may represent a total width of 6 to 12 ft., the intervening country has been found to measure up to 40 ft. Where a lode is thus bifurcated each branch has its pay shoot or shoots. Regarding the size of these quartz reefs, they are found to range from 1 to 15 ft., only three of them being really strong bodies.

Each vein seems to have some peculiarity of its own, and the irregularity of two of these deposits is still further accentuated by a sudden flattening of the contact joint at a vertical depth near the middle of the property of about 500 ft. Towards the south-east this level country commences just below the 400 level; to the north-west at a depth of 900 ft. it has not yet been met with, and, although still incompletely proved, it does not appear to exist there. In spite of this change taking place in a few feet it cannot be termed an abrupt one. This flat ground extends towards the north-east for about 400 ft. with only a fall of 16 ft., or 1 in 25. Then follows a sudden recovery, and for 200 ft. the contact joint is nearly vertical; below that again the normal angle, 60°, is gradually regained.

Not only has this unexpected change upset all calculations in laying out the mine, as will be seen later on, but, to judge from a remarkable feature recorded below, the ore bodies themselves, when in course of forming, seem to have been equally unaware that a sudden alteration in the angle of the contact was about to take place at this point. Unpractical as such a statement may appear, this is undoubtedly the impression received from working the ground, and an end section of the Homeward Bound vein at this point serves to illustrate what is now described. Below the 500 level this vein, after striking the contact joint (here quite vertical) at the usual angle, continues for some distance along it. In follow-

ing up this ore body it was found that it did not cease or turn over when the flat ground was reached, but went on up as if it had accidentally shot past the shoulder in the joint forcing itself into the overlying serpentine. The lode at leaving the diorite was about 18 ft. wide, and tapers upwards until it gradually cuts out, the apex being some 20 to 30 ft. above the shoulder. True serpentine walls enclose this body, which was found to exist along a length of 140 ft. The lode at this point is highly auriferous.

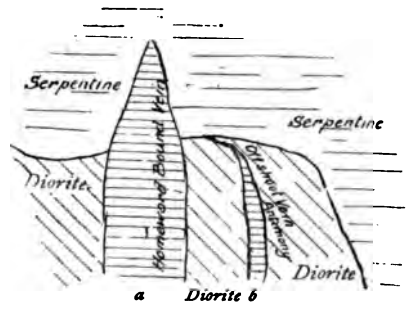


FIG. 3.

Section showing Homeward Bound vein previous to its striking the contact-joint. *a*, main vein; *b*, offshoot, carrying antimony.

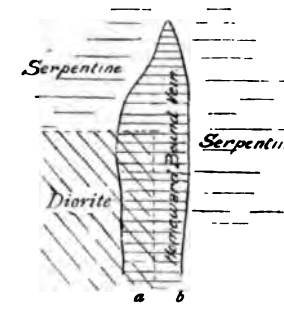


FIG. 4.

Section showing same vein after it strikes contact-joint. *a*, small pay seams strung through "diorite" rock; *b*, veinstone proper.

From the above, the writer does not intend to give it as his opinion that these ore bodies received their gold from any deep seated origin, but rather to illustrate a very extraordinary and interesting feature of the mine, and one that has to be accounted for in advancing any theory as to the deposition of the gold in its present state.

Barren zones occur in all the veins, rendering them valueless over a vertical height ranging, so far as is yet known, from 40 to 450 ft. Sometimes the lode remains strong throughout, sometimes it cuts out completely, leaving little or no clue but the known general direction to enable it to be searched for and found again. Examples of such are seen in the Shamrock group, where the veins disappear entirely for 450 ft.; and in the Welcome, where the reef remains strong, but absolutely valueless for 130 ft. This latter re-makes into an exceedingly rich lode, but lower down becomes once more unpayable. It is noteworthy that this last change occurs where the serpentine is pinched and ill defined, the two diorites being but a few feet apart.

The smaller veins are very eccentric, there being often no quartz

reef bringing them to the contact, or perhaps only a seam a few inches thick. Where such is the case, bunches of rich ore are frequently found, forming a broken line, usually just in the diorite, or, more rarely, actually buried in the serpentine; these are locally termed pipe veins.

Two ore shoots have been found at the N.W. end of the ground, neither of which can be traced above the 470 ft. vertical; one of these cuts out again after a short career of 50 ft. The length of the ore shoot in this latter case at no time exceeded 5 ft., and its width less than 2 ft., and yet the value of this insignificant vein was no less than £5,000. The other which was payable to the 750, broke up at the 470 into rounded blocks of exceedingly rich arsenical pyrites; they ranged from the size of a man's fist up to a length of 9 ft. These blocks of ore were found buried in the serpentine abutting on to the diorite, and were separated from one another by several feet of barren ground; no quartz or other gangue was present where this vein cut out, though the reef 50 ft. lower down is a strong one.

Further along in this direction (N.W.) the serpentine pinches to only 2 or 3 ft., and is not well defined. Two veins about 2 ft. wide have come up to the contact joint, but although showing calcite and all the other characteristics of the Lucknow ore bodies, they contain no payable gold. The ground has been prospected for many hundred feet beyond the last pay vein.

Short breaks running N.S. are found along the contact, usually occurring where the pay veins strike it: with one exception, these do not throw more than 10 to 12 ft. This exception is in the case of the important reef, the "Welcome," which actually changes the direction of the contact joint no less than 70°, and for over a distance of nearly 500 ft. The ore shoots belonging to this vein were three in number, and of immense value; they lay along the north side of the reef, between it and the serpentine. At the 500 level, one of these shoots falls away into a "dropper." This "dropper" cuts out at the 600 level, where it is already distant nearly 50 ft. from the main reef.

Before passing on to describe the mining methods adopted, certain features of the country containing these veins demand some little attention. Two geological pecularities tend to make prospecting a work of great difficulty. The first is the presence of detached boulders of diorite buried in the serpentine, sometimes leaving 2 or 3 ft., sometimes only a few inches of serpentine between it and the main body of diorite, and in certain cases being almost welded on to the latter. These boulders vary in size from a few

inches up to masses of 100 ft. in diameter. Some of them remain unchanged in character from the original stone, whilst others are so metamorphosed as to be unrecognisable. When in the latter state it is known among the miners as "old man" rock, on account of its excessive hardness and toughness. This latter is often found extending for considerable distances, but only ranging from 1 to 5 ft. in thickness, and lying between the diorite and the serpentine. One class of this stone, on a analysis being made, gave 89 per cent. of silica, but the normal type will be found to yield 76 per cent. Another class is composed very largely of asbestos, greatly adding to the difficulty of advancing through it.

The other feature is that known on the mine as "false joints." These are breaks in the diorite running almost parallel to the contact joint from which they start, both in the horizontal and vertical line. A soft country or "dig," often accompanied by calcite, is found along these breaks, and they therefore closely resemble the true joint, for which they have many times been mistaken.

In the work of following the joint two errors are exceedingly easy to make, even when the greatest care and vigilance are being exercised. The one is mistaking of "boulder" for the main body of diorite, and driving round the same; the other (a fault which is always made in endeavouring to guard against falling into the first error) finding a break in the diorite, and concluding that it is the contact joint. The greatest difficulty in determining the true position of the country is experienced in cross-cutting: (1) if when driving through the serpentine to strike the diorite hard stone is struck, there is no proof whether it is the true diorite, or whether it must be passed through in order to reach the main body; (2) if, on the other hand, a cross-cut from the diorite side is being put in and a false joint is struck, the "dig" lying along it so resembles the serpentine, in the altered condition in which it appears when intervening between the main body and an extensive boulder of diorite, that it is repeatedly accepted as such, and driving along it is commenced in search of gold-bearing veins.

Two minor points may also be remarked on, which, although only causing temporary checks in prospecting, add considerable local colour and are distinctly interesting from a geological point of view. One of these is found in the occasional absence of a sharply defined line between the diorite and the serpentine. Usually where boulders are absent no trouble is experienced in at once determining the two stones, but regions exist where no demarcation can be traced. At such places the diorite passes gradually and imperceptibly into serpentine, this change often extending over several feet of width. About the middle of this

undefined country the rock generally takes the dark green or blue-black colour of the serpentine, but close inspection shows the crystals of the diorite. This class of stone is known locally as "black rock," the true serpentine being designated "blue" and the diorite "grey."

The other point worthy of notice is that along certain stretches of contact "serpentine" faces are found to have penetrated back for several feet into the diorite. These faces must have been formed by friction, but inasmuch as they are found on the outside of and often in a boulder, great uneasiness is experienced when a drive in what has been taken as diorite suddenly comes upon them. Not unfrequently such doubts are groundless, but occasionally these faces do a good service, and cause an error to be corrected. Where most reliance is placed in determining the ground is in the jointing, that of the main body of diorite being angular and parallel, that of boulders being for the most part rounded; but this is not always infallible. Sometimes the very large boulders retain their jointing, and more or less their direction of jointing, whilst, on the other hand, the writer has found not only rounded surfaces 15 to 20 ft. back into the diorite, but distinct and separate little boulders, giving the appearance of being welded into the main body.

The peculiarities described above and the consequent mistakes arising from the same have been mentioned at some length as being both of geological and economic importance. *Unless the true contact joint be followed payable ore bodies must be missed*, and the abnormal richness of these render the work compulsory at almost any cost. Fortunately these difficulties do not exist throughout the length of the property, several hundred feet of straight and well-defined joint existing between this broken country.

MINERALOGY.

The minerals found on this field are not numerous but are exceedingly interesting. They may be at once separated into two classes—(1) payable, (2) non-payable.

(1) Under the former are ranged—free gold, auriferous arsenical pyrites, and antimony carrying gold.

(2) Under the latter are found—magnetic pyrites (pyrrhotite), common iron pyrites, zinc blende, and more rarely galena, common pyrites, stibnite, kermesite, and cobalt-bloom.

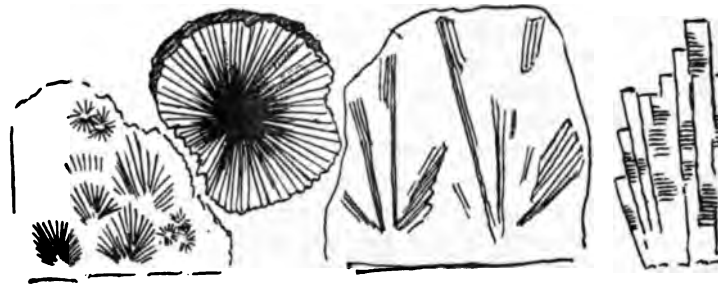
(1) *Free gold* occurs in exceedingly rich patches in the calcite; when in this very rich form it is most often dendritic in structure, but it is also found tabular or threaded through the stone. Where

the rock is less rich the gold differs but little from the ordinary reef gold, although usually met with in calcite. It is seldom found very massive, a 1-dwt. piece being considered large. Pieces of stone several pounds in weight have been and are still being found, yielding up to 40 per cent. bullion. Such stone on being treated with acid gives an excellent illustration of how the gold exists in the rock, for the matrix being dissolved, the moss-like form of the gold is apparent. Rich stones of this class are not by any means regarded as office specimens, for they are not unfrequent, and are treated with other ore going from 50 oz. and upwards; the output of the mine depends very largely upon this "special" class of ore.

Up till four years ago, that is to say, while oxidised zones were still met with, large cavities or "vugs" were found in the larger quartz veins; in these there existed a quantity of brown matter, sometimes of an earthy, sometimes of a spongy nature, and this carried as high as 50 per cent. of gold; by washing and squeezing, the valuable contents were easily extracted. It was mainly to the finding of such pockets that the phenomenal returns of that time were attributable; these were 16,345 oz. in March, and 16,965 oz. in April, 1895, at that time a record yield. Unfortunately no specimens of this so-called "brown vein" have been kept.

The value of the bullion is £3 10*s.* an oz.

Arsenical Pyrites.—This mineral is the most important in the mine; and since the oxidised zone has been passed through, more gold exists in this pyrites than occurs in a free state. Arsenical pyrites is found both in radiating and massive form, though the latter by a play of light very often shows a divergent structure. The radiating form is very beautiful, and is supposed to be unique. Excellent specimens are shown of this stellate structure, locally it is termed "Prince of Wales's feather" pyrites. These forms usually have from ¼ to 1 in. radius, but portions of considerably larger bodies have been found.



Figs. 5, 6, 7, 8.—Mispickel in carbonate of lime.

Careful analysis to detect the presence of antimony sulphide, and so account for this radiating form, have failed to show that mineral. The arsenical pyrites assays very high for gold, the clean mineral in which no gold can be seen yielding 100 to 800 oz. per ton, and from other examples in which gold can be seen coating the faces, sticking out from or otherwise closely associated with the mispickel, assay results of 1,500 to 3,000 oz. to the ton have been obtained from lumps of considerable size.

Blocks of solid mispickel weighing from 2 to 5 tons have been met with during the past three years, which taken intact would assay about 250 oz. of bullion to the ton, and in some instances might have given better results but for the presence of pyrrhotite. For some time it was considered that the arsenical pyrites in which the radiating structure was displayed, was richer than the more massive, but it is not so. Not the least interesting of the specimens shown, is an exceedingly fine grained arsenical pyrites occurring in a clean pearly lustred calcite in which the cleavage is well determined, such stone, although not carrying more than 25 to 30 per cent. of mineral, assays from 40 to 50 oz. of bullion per ton. A distinctly different class is found in a dense fine grained massive stone having an almost rectangular cleavage, but otherwise highly uninteresting in appearance; a portion of this stone assayed 321 oz. bullion. A far richer stone is found in specimen No. 3. A photograph of a very remarkable specimen is also shown, this being no less than a boulder of arsenical pyrites highly auriferous (about 2,000 oz. to the ton), which was found buried in the serpentine where a vein cut out another boulder very similar to this, but weighing about 2 tons, found close to it.

Antimony.—This is more erratic in its occurrence than the other minerals, and besides being auriferous it almost invariably betokens the vicinity of rich arsenical pyrites. The value of the antimony lies entirely in the gold contents, and this varies somewhat. There may be said to be two classes of this mineral: the one is hard and untarnished and grouped in heavy well defined crystals with striated faces; these crystals are found shot about in calcite. This form of antimony is not rich in gold, yielding not more than 1 to 3 dwt. gold per unit of antimony. The other class shows a dense crystalline mass and the mineral being found in splashes or bunches it does not appear to carry any quartz or calcite matrix. It is much softer than the first mentioned and, certain faces being tarnished black, it is evidently accompanied by stibnite. This class of mineral carries considerable gold, but varies with the locality in which it is found; it assays not less than 6 and more than 10 dwt. per unit of antimony. Bullion won from the anti-

mony ores is good, far better than than obtained either in a free state or in combination with the arsenical pyrites. As has been remarked, antimony is a good indicator, and several valuable bunches of high grade ore have been found by paying attention to a few specks or a tiny seam of antimony.

(2) *Non-payable minerals.*—Magnetic pyrites, at once the most prominent and the most unsatisfactory feature of the mine. Pyrrhotite occurs in zones in the veins, and either entirely replaces the auriferous mispickel or is so intimately associated with it as to almost smother it, bringing the value of the lode matter down till it verges on the unpayable. The magnetic pyrites seldom occurs massive, it being more often spotted about or displaying a reticulate form. Splendid pay bodies have gradually given place to this unwelcome mineral, which lives both in the quartz and in the calcite. Not unfrequently, after 50 or 100 ft. of worthless pyrrhotite, the arsenical pyrites re-asserts itself. Where the magnetic pyrites is strong, great uncertainty exists as to whether certain blocks of ground will repay extraction and treatment. Frequently each sample contradicts the previous one; carefully taken daily samples from a drive advancing along a strong lode, highly mineralized with pyrrhotite, gave the following sequence :—6 dwt., nil, 22 dwt., nil, nil, 3 dwt., nil, 12 dwt., nil, trace, results that hardly tend to elucidate matters. Where a strong calcite lode carrying pyrrhotite is found, it has to be subjected to very extensive prospecting in order to show whether parts of it may not prove payable. The pyrrhotite carries no nickel.

Iron pyrites and also *zinc blende* are found chiefly in the strong quartz reefs and apart from the gold-bearing calcite. This however is not a hard and fast rule, and free gold has been found on the field in conjunction with these minerals.

Stibnite (sulphide of antimony).—A sample of this is shown. Although classed as unpayable it is nevertheless auriferous, and the clean mineral carries about 8 dwt. of gold. It is occasionally found with the native antimony, but in no quantity, and is therefore unimportant.

Kermesite and Cobalt-bloom.—These are of rare occurrence; the stone in which they are found has always been a good payable calcite carrying mispickel, native antimony, and free gold.

MINING.

Prospecting. From the descriptions given of the Lucknow ore bodies, it will be readily understood that the work of following them is not by any means a simple task. The veins in themselves

are sufficiently eccentric, and unreliable to demand constant and close attention, and the work has been rendered even more difficult by reason of the irregular formation of the containing country. Again, the knowledge that a pocket of rich ore may lie a foot to the right or left of existing workings, giving no indication whatever of its presence, adds an element of sustained uncertainty, and it may be truthfully said of the engineer in charge, that he can never know when to leave off prospecting in any one place.

Another point that will at once be recognised is that, robbed of all these weaknesses and given veins of comparative continuity and regularity, there must necessarily be two plans by which to work them, extending as they do over a horizontal distance of 3,000 ft. One of these plans is to sink either shafts or series of winzes connected by short drives, to follow down each payable vein, and to drive connecting levels, say every 300 to 400 ft. of vertical height, for purposes of ventilation, supervision, &c. These shafts would have to dip both with the contact and with the reef that carries the ore shoot, and they could only be fitted with skips. By adopting this method all idea of discovering other ore shoots lying between the known veins (ore shoots not reaching to the surface) is at once dismissed. The other plan is from conveniently placed shafts to put in drives at different levels along the contact joint, with the intention of uncovering both known and unknown pay bodies that may be found to lie within the auriferous zone. Briefly it was a question of extensive shaft sinking in very hard stone as against constant driving, taking advantage of the soft serpentine. Apart from the peculiarities of the ground, both these plans had great disadvantages, and each must necessarily prove very costly.

Unfortunately, as on all but the new fields, the setting out of the mine had to depend very largely on former work, and could not well be altered without the expenditure of large sums of money. Where a big and proved reef is in question, and where bores can be put down, there can be little hesitation as to whether certain work is justifiable or not, but in a pockety mine such as that now under consideration the case is very different.

The early history of the field may be briefly epitomised thus. For 40 years previous to the formation of the present companies, the field had yielded immense quantities of gold. At first the individual miner worked the gold shed from the outcropping bodies; then—when the sources were tapped—parties of men followed them down; later on, when water began to be troublesome, many small local companies acquired and worked each separate lode. The following extract from the Mining Warden's Report to the Government gives a

fair idea of the immense value obtained from the shallow workings of the field. The extract is taken from the report dated 1884, but the warden alludes to a previous period. "In the early days many rich veins were left in the hard ground, and have not been seen since. The original Golden Point vein yielded to the shareholders the enormous sum of £50,000 in nine months; Spicer's, 80 ft., £42,000 in six months; Golden Gate, 80 ft., £35,000 in eight months; Shamrock, 80 ft., £30,000 in two years; Crinoline, 20 ft. of ground, £20,000 in 18 months; Snobs, 80 ft., £20,000 within one year; besides several smaller returns, making in all a large round sum within two years. The pyrites veins have not yet been opened up." A weak zone occurring in several of the ore bodies, and none of the enormous profits having been reserved for such a contingency, these syndicates were unable to do the dead work thus imposed on them, and only desultory operations were carried on until, in 1890, their various properties were bought up, amalgamated, and re-sold for a large figure to the Wentworth G. F. P. Co., which was formed to mine the whole field. Subsequently the Aladdin's Lamp Gold Mining Company was started as a subsidiary enterprise, with a lease of proved ground lying north-west along the contact joint. This latter company has, since its formation in 1892, paid some 200 per cent. in dividends. With one exception the veins were both poor and weak at the time when the Wentworth Company took over the property, and very extensive prospecting was at once necessary.

Now whilst in the lower levels only certain pay shoots are found, and a great extent of non-auriferous country lies between them, such was not the case in the upper levels, where comparatively short spaces intervened between the more numerous ore bodies. Thus driving along the contact was the quickest and most convenient way of finding and working the lodes. Lower down this system was continued, in the hope that the pinched veins would re-make, so that this method, involving extensive driving, was adopted; and whilst it incurred an immense amount of dead work, it held out reasonable chance of recompense for the same.

In pursuance of this policy levels have been driven along the joint at every 100 ft. vertical. These are put in through serpentine, the foot wall of diorite being only just broken, and the stone exposed, in order to make sure of disclosing any quartz or calcite lode or series of seams that might, when followed into the diorite, be found to form into a lode. It is customary whenever any vein or promising spot is reached not to delay the main drive but to push on, other men being kept following up behind prospecting these likely looking places. Whenever a point lying in the line of direc-

tion of a known vein is reached and no trace of it can be seen, very thorough search is made, a rise is put up, a winze is sunk, and intermediate levels are driven from each before it is deemed justifiable to abandon hope of finding the missing stone. A diamond drill is frequently used to assist in this. With these drives constantly going ahead, between a mile and a mile and a half of prospecting has been done each year by the two companies combined. At a depth of only 800 to 900 ft. about 10 miles of work had been put in. The extreme horizontal distance explored over is 3,500 ft. Finds at the north-west end of the ground have justified the system adopted.

The cost of driving in the serpentine along a good joint breaking the diorite foot wall is about 24*s.* a foot, hand labour, sometimes falling as low as 17*s.* 6*d.* In driving through diorite air drills are generally employed, the cost of passing through ordinary rock of this class being about 30*s.* a foot. On the other hand, where the "old man" rock lies against the diorite short distances costing up to £5 a foot with machine drills have had to be traversed. Prices range between these two extremes. As an example of the difficulty experienced when the very hard and tough rock is encountered, cases may be cited where it has taken as much as two shifts to get down a hole of 30 in., hundreds of drills being blunted. With a machine at work on similar stone, 10 drills to an inch of progress may have to be used at starting, and often a hole has to be abandoned and another commenced.

Very carefully compiled tables showing cost of work are always rendered with the reports of the two companies, but any comparison between working expenses here and elsewhere is quite impossible. Those conversant with mining need no further proof of this statement than a bare description of the field; there are others, and very naturally they include many shareholders, who do not appreciate the peculiar nature of the Lucknow deposits, and would wish to compare cost with that involved in working the immense oxidised ore bodies at Kalgoorlie, or with the certain deposits of the Rand. The details given above, however, show how different the conditions are at Lucknow, and how useless any such comparison would be.

At a depth of 700 ft. only five of the original pay shoots were found, but as one of these had pinched out for over 400 ft., it is not improbable that other revivals will be found to occur as depth is attained; this reasonable hope has given great encouragement for systematic prospecting. No better example showing how these rich veins quickly pay for extensive search work can be found than the one just referred to, namely, the Shamrock. In this case the cost of a drive 400 yds. in length (a work occupying nearly a year with

hand labour) was almost won back during the first month after the object of the drive had been attained, although only two small broken seams or veins were uncovered, yielding little or no battery stone.

Stoping.—A very few words will suffice in which to dismiss this subject, for the conditions are so different from those of other mines, and the rock is so hard, that the usually prescribed methods of blocking out and extracting pay ore have frequently to be set aside. Here, again, figures with reference to cost of stoping are of no value for comparison, for when an ore shoot has been tracked, almost as much trouble ensues in working as has been experienced in finding it; this is, of course, due to its erratic nature. When possible, the usual methods for stoping are adopted; but these need hardly be referred to. When an ore body lies between the diorite and the serpentine, the latter, when exposed more than a few feet, becomes very treacherous, flaking off in large blocks; overhand stoping is, therefore, compulsory, and the depleted parts are run in with mullock at stated intervals, the miners being in the meanwhile transferred to another stope. Repeatedly a strong, healthy-looking body of pay-stone has been struck, and work at 50 or 100 ft. lower has been put in to strike the same, and to allow both overhand stoping and a proper get-away for milling dirt; more often than otherwise, however, no trace can be found of the ore body. Later on, on following down the stone, it has generally been found to have split into several seams, often of rich ore, separated by many feet of absolutely barren diorite. If the ground were soft, the whole lot would repay removal, but this class of rock—influenced by the presence of fine grains of pyrrhotite and a very large percentage of silica, is such that it allows of little more than 4 ft. of boring per shift. These seams sometimes make into small bunches of exceptionally rich ore, and the only safe rule to carry out in order to avoid missing these little bonanzas is to follow anything and everything that returns more than 3 dwt. of gold. Again, some of the "pipe veins," although exceedingly rich, do not extend beyond the length of the rise or winze that is put to follow them, so that no stoping ground is opened up. Thus it follows that there is often but a few weeks, and seldom more than four or five months' stoping ground in sight, in spite of the energetic and ceaseless prospecting work that is being carried on. All boring is done with the double hammer.

The Shafts.—Four are sunk on the property, besides other unimportant and abandoned ones. Two of these are little used now, the *Jackass*, at the N.W. end of the ground and in the Aladdin Com-

pany's property, ceased to be of use after that Company's old battery, which stood on this site, was pulled down; this shaft has a depth of 600 ft. The other is the *Reform*, a small three compartment shaft reaching to the 400, whence an inclined winze carries communication down to the 600 level (this latter is sunk not on the true but on a false joint which drops away from the real contact about 50 ft. below the 400). This shaft is near to the battery, and has all the shops grouped around it. It is fitted with a Cornish pump which raises water from the 300 level along which the main drain runs. Incidentally, it may be mentioned that this shaft came in very useful during the recent labour trouble, when ore could not be raised from the other shafts owing to intimidation, and the staff, together with the few men who remained loyal, were able to raise by means of winches underground and of the *Reform* winding engine, sufficient stone to keep the battery running for the 13 weeks during which the strike lasted. This not only greatly discomfited the strikers, who were unaware that a reserve of broken ore was on hand, but also allowed the monthly gold returns to be maintained, so that nothing was lost to the Company.

The two most important shafts are the *New Main* and the *Phœnix*, situated at the S.E. end of the property. The New Main is a well constructed work 890 ft. in depth. This shaft was sunk with the object of cutting the contact joint at about 750 ft., but owing to the flat country already referred to, the diorite was struck near the 500 level. As a further illustration of this flattening of the joint, it may here be mentioned that at the 400 level there is a cross-cut to the south 360 ft. in length to reach the contact, whilst on the 500 a cross-cut north for 40 ft. was necessary to attain the same object. This unexpected development was more far-reaching in its results than would be supposed, owing to a company having been formed to work the veins at greater depth, after that they should have passed the Wentworth Company's property. However, as has been recorded, a recovery took place so sudden and complete as to bring the contact joint at the 800 into a position not far from that which it would have occupied had the normal inclination of about 60° been maintained from the surface down to this level. From the 500 to the 700 the joint is not less than 80°, and from there on to the 1,000 ft., which is reached by a winze, some 65° is averaged. Thus the shaft which at one time appeared to have been thrown away is well placed, and on the whole could not be bettered either for prospecting or extracting purposes.

The other shaft, the *Phœnix*, is situated between the *Reform* and the *Jackass*, and was until lately a vertical one for 150 ft., and then,

the contact having been struck, the shaft was carried down on it at an angle of 40° out of the horizontal. Between the 300 and 400 levels the angle of the joint increased, and from the 500 to the 700 it is almost vertical. No alteration, however, was made in the dip of the shaft, the reason, without doubt, being that those responsible for the work were misled by the immense boulders of diorite lying against the joint, and whose outer edge lay parallel to the true contact as found in the upper levels. Thus much of the shaft was sunk in or upon the inclined edge of these boulders, which must have been mistaken for the main body. At 700 ft. a diamond bore-hole was put back into the supposed diorite to see if serpentine was to be found, but 90 ft. exposed no trace of any (the joint was afterwards found 310 ft. distant to the south, the same direction that the diamond bore was given). Shortly after the writer took charge of the mine workings, the shaft, which was worked by a single skip—the other compartment being laddered—was greatly improved. A pillar was left just above where the straight shaft struck the contact, and a start was made to continue the underlay to the surface; at the same time, sinking to meet this was begun from grass. The distance was 257 ft., and the work was completed in less than three months. Meanwhile, head gear, ore bins, engine and boiler foundations, &c., were prepared, and, when ready, hauling by the old shaft, which had up till then been carried on without interruption, was stopped, the engine and boilers were thrown back to their new position, the pillar at the 150 was cut away, and in a very short time the new installation was running, almost without a break in the supply of milling dirt from that end of the mine. The shaft then worked two cages, instead of a single skip, and two large man-cages (put on before change of shift); these latter placed the men at their work 15 to 20 minutes sooner than the old method. Immense economies in wages, in fuel, in the life of the ropes, and in many other points have been effected by these alterations. Both the Wentworth and the Aladdin Companies use this shaft, and each Company has a 150-ton ore bin at the surface built into the poppet legs.

The working arrangements as now carried out are as follows:—Into the ordinary cage a truck is run and secured; at each level there are balanced doors, which when closed down allow the cages to rest against them. These doors are always open, excepting at that particular level at which trucking is being carried on. When work is completed at that plat the doors are again left open and the gangway is clear. The driver at the surface can, by means of using his signal bell, ascertain whether the line is clear. The man-cages

when not in use are simply lowered about 20 ft. below the bottom level; a timber cage is also kept there. The time required to change these cages is very short.

A platman on each shift controls the doors, lands the trucks at the various levels, changes the cages, and has charge of the signals, telephone, &c. This arrangement has now been working for three years, and no accident either to life, limb, or machinery has ever taken place.

The man-cages and safety-grips attached to them were designed by the writer.

A very complete system of air pipes runs through the workings, many miles of them being laid down. Use is made of air power on every available opportunity. Air winches are used to assist in the sinking of the many winzes; these are moved from place to place. More powerful winches are fixed permanently to raise battery dirt from pay bodies lying remote from the shafts. One of the most important winzes lies between the *Phœnix* shaft and the *Jackass*, and connects the 300 with the 700 and intermediate levels. This is a vertical shaft, and is fitted with a full sized cage; a strong engine stands on the 300 level. Elsewhere in the mine are other important winzes.

Air pumps are numerous, and work is seldom if ever delayed by any trouble with water. The mine water is raised to the 300 level by means of air pumps, whence it is lifted direct to a reservoir on the hill above the battery by means of a Cornish pump.

Several small fans to ventilate the far workings are employed; these are moved from place to place as required. They are worked from engines driven by compressed air (sufficient margin was not allowed when the electric installation was put up to permit electricity to do this work; difficulty of ventilation did not at that time exist).

Rock drills are used where practicable, and the air mains are tapped where necessary to allow the diamond drill to prospect the country.

Treatment of Ore.

The fact that the two companies with a combined capital of £600,000 have only 35 head of stamps between them marks the undertaking as being of unusual character, and this, together with what has been said of the richness of the ore, clearly prepares those who have followed this paper thus far for a system of ore treatment, both above and below ground, differing in certain points from that found at most mines.

The immense richness of the gold-bearing stone is at once a source of strength and of weakness to the companies. The latter is found in the very necessary precautions that have to be taken to prevent stealing, a trouble that has on occasions been very rife and which, in spite of every care and watchfulness, it is almost impossible to entirely eliminate. A paper might be devoted to this subject alone and some exciting experiences recounted.

Underground. "Firsts" or Special Ore.—When rich ore is struck it is at once reported to the management under pain of dismissal. A member of the staff then becomes responsible that all good stone is bagged in the rough and placed under lock and key in a sorting room under ground; no shots are fired in rich stone without his knowledge, and he or his assistant is on the spot before the men return to the face. In the sorting or picking room trusted men are employed to spall the stone and to make "firsts" of it, detaching as much of the gangue, or possibly country rock, as possible. The picked stone is filled into numbered bags, made up to about 60 lbs. weight, which are then sewn up. This picking only takes place on day shift, and at the end of the shift these bags are weighed and their numbers are taken by the member of the staff who is responsible; they are then passed in to an inner locker or strong room. There being no grade between this and the milling stone, the balance from the picking room is passed away as ordinary "seconds," or comes in usefully to sweeten up any poor patch of milling dirt. There is a sorting room and a strong room at each end of the Wentworth ground and the Aladdin Company have also their "lockers."

Ordinary Milling Dirt or "Seconds."—The breaking and tramming of this offers no special features. One point, however, may here be noted, namely, that owing to the smallness of the veins it almost invariably becomes necessary to "cull" over all the broken "seconds" and to pick out the country rock which coming away with the lode matter seriously reduces the value of the battery stone if it is not thrown on one side; this is generally done in the stopes.

Surface. Ordinary Milling Dirt or "Seconds."—Up to three years ago the batteries of the two companies were separate, but about that time, that of the Aladdin being in a tumble down condition, the Wentworth Company added on an extension to their own mill. This arrangement proved to be immensely advantageous and ecomomical to both companies, for the same engine now works all the stamps, the same motor runs the vanners. Further, and more important still, one staff of battery hands controls the two mills—thus effecting a saving of 50 per cent. in the heaviest item—and better supervision is now possible. The main engine separates the two mills and no confusion

is possible. The parent company charge the Aladdin so much per ton for crushing. Milling stone is run along the tramways both from the main and Phœnix shafts to the battery; the distance in each case is not great.

The battery is by no means a model one, but it does excellent work and is well run. The rock breakers are placed on bed rock, and the stone after passing through them is raised by a hoist to the top of the battery building and tipped into the bins behind the stampers. The dirt is thence fed into the mortar boxes by an automatic feed—the Templeton roller principle is used with remarkable success. Inside amalgamation is gained by means of housing plates and by dropping small measures of quicksilver into the mortars at intervals during the shift: this measure varies from 2 to 4 grams per hour to each five head battery. An attempt was made to introduce a back splash plate, but, owing to the hardness of the rock, it was cut to pieces after a few days run. 850 lbs. stamps are used, a weight that is hardly sufficient to cope with the exceeding toughness of some of the rock; but as a greater weight would slime the less hard and generally richer stone and give an even larger percentage of fine pulp in the tailing than has now to be dealt with (*see* Treatment of tailings), the employment of heavier stamps would mean an increased cost in the cyaniding. Interesting experiments have been carried on to find out the best metal to use for shoes and dies, but the results have been very conflicting. Krupp's chrome steel, Hadfield's manganese as well as chrome steel have been used, besides ordinary cast iron; these have worked on both forged steel and cast iron dies. The subject, however, is outside the province of this paper and may possibly be dealt with separately. No. 7 needle angle slotted screens, front discharge, are used, and the pulp splashing through these travels over usual copper plates; the tables are 16 ft. × 4½ ft., the newer ones having two 3-in. besides the lip plate drip.

From the silver trap at the foot of these the pulp passes direct to Frue vanners, two vanners being provided for each five head battery. The value of the pulp as it passes on to the vanners ranges from ½ to 3 oz. bullion, and the concentrates vary accordingly. Where much pyrrhotite or mundic is carried, these latter may only yield 6 to 8 oz. bullion per ton; when free from these minerals, the arsenical pyrites caught on the vanners run up to as high a value as 60 and even 100 oz. bullion per ton—a single example having been recorded in the Aladdin battery giving 172 oz., the sampling extending over 24 hours. As the tailings from such rich pulp necessarily run high in gold after leaving the vanners, it is the endeavour

of the mine management not to send up stone of this class, but to keep a fairly uniform value of about 20 to 25 oz. bullion in the concentrates; this task, however, is an exceedingly difficult one, owing to the peculiar characteristics of the mine, and the fluctuations are constant. Daily assays are taken both of the concentrates and of the tailings, the sample from the latter being obtained by means of an ingenious automatic contrivance that every 40 minutes momentarily diverts the tailings stream into a receptacle. The vanners do excellent work, and perhaps no battery pulp in the world serves better to illustrate the remarkable value of these machines. The speed at which they travel necessarily varies, averaging perhaps some 5 ft., and the side shake running from 150 to 200 per minute. The following are taken from the assay record of the two companies and are fair examples of the excellent results obtained; no assay was taken of the pulp as it left the copper plates, but they average some 3 per cent. of mineral.

Wentworth concentrates ..	21·20	22·80	24·20	7·00	8·40 oz.
„ tailings	0·20	0·20	0·20	trace	0·10 „
Aladdin concentrates	21·80	16·60	27·80	34·30	66·00 „
„ tailings.....	0·20	0·15	0·25	0·25	0·40 „
Wentworth concentrates ..	42·40	13·60	40·70	40·20	31·00 oz.
„ tailings	0·40	0·10	0·45	0·40	0·25 „
Aladdin concentrates	35·40	14·04	44·50	19·00	22·20 „
„ tailings	0·35	0·10	0·50	0·175	0·15 „

The concentrates are dried in a room, the sheet iron floor of which is heated by exhaust steam from the battery engine; they are then bagged and sent off to the best market. Until lately all this was shipped to England; latterly, however, the Smelting Company of Australia have secured the purchase and the ore is treated at their Dapto works. When these concentrates were shipped the filled bag was sewn inside another; now that only one handling is necessary the outer bag is dispensed with.

Tailings.—The tailings are elevated by means of a bucket wheel, and crossing the main road through a 6 in. pipe are dumped in such a manner that the water drains off and is re-used. The accumulated tailings are very valuable. Cyanide works have not yet been installed, but have, I understand, been commenced; very great difficulty has been found in designing a plant to deal with the slimes, which, owing to the large percentage of carbonate of lime, exist to the extent of 60 to 70 per cent.; these slimes carry the greater value.

"*Firsts*" *or Special Ore.*—For many years the mode adopted in the treatment of this rich stone differed greatly from the present

plan. The original custom was to put the stone through a small breaker, and then, without further treatment, to fill it into barrels and ship to England, where eager competitors to secure the same ensured very favourable terms. Latterly, however, the ore is partially treated on the spot in order to secure the free gold, the interest on which while it was afloat had been hitherto lost. Another advantage accruing to this change was that sampling, which had been both difficult and at best only yielding approximate results, was greatly facilitated and rendered as near accurate as possible.

The present mode of work is as follows:—Once or twice during the month a lot, comprising 1 or 2 tons, is taken from the locker below ground and sent up, each bag being re-weighed immediately before its contents are emptied into the hopper. A battery of five head is set aside to treat this special ore, and the pulp, after passing over the copper plates, is diverted into settling tanks and is not run over the vanners. If the stone is rich in free gold it may happen that the plates must be cleaned up every 10 to 20 minutes. A member of the staff is always present when this is being run through. When the tailings have settled in the tanks the water is siphoned off, and the pulp is dried and bagged, or casked, in the same manner that the concentrates are treated. Generally a stone rich in free gold will not give a very valuable residue, and *vice versâ*, but this is not by any means invariable. Most lots of special ore after yielding 100 to 120 oz. of bullion to the ton will contain 150 oz. of fine gold in the residual pulp. Where calcite forms the matrix considerable loss in weight follows this treatment, as the calcite forms slimes which float off; they, however, carry only a few dwts. of gold with them, and, joining the battery tailings, are not lost.

Sampling.—Necessarily the sampling for market purposes of such rich stuff as that obtained from the "firsts," and also of the higher grade of concentrates, has to be carried out with great care. For some time the custom was to run all the dried ore into a small hopper leading into a short down pipe which was subdivided many times until $\frac{1}{16}$th of the bulk fell into a receptacle, out of which a sample was taken, the rest falling into a large hopper, from which the bags or casks were filled. Six check assays were made of the sample thus obtained, and the mean of these was taken. Later on it was found that, however dry the ore was and however carefully mixed, some of the richer particles adhered to one another, forming small lumps, and that these seldom passed through into the sample box; the assays therefore under-estimated the true value of the parcels. In place of this sampler a gauge like an elongated cheese

taster is now used, and by the aid of this cores can be obtained from the filled bags which give far more reliable returns.

Treatment of Antimony-carrying Ores.—With regard to the effect on the copper plates of antimony contained in certain of the stone sent to the mill, a few interesting points may be noted. This metal is not found to sicken the plates to any appreciable extent, but the amalgam collected when much antimony is being put through is low in quality, averaging only about 33 per cent., instead of 45 to 47 per cent. bullion gained from the usual amalgam. When low grade amalgam is obtained it is generally cleaned, the dross being floated off; this latter is found to contain about 17 per cent. of high grade bullion, which is recovered by roasting the dross before retorting.

A curious specimen is exhibited, consisting of a piece of stone carrying about 75 per cent. of pure antimony. This stone had got among the "firsts," and was found in the mortar box after a parcel had been put through. It shows the outside faces to have taken up quicksilver and formed an antimony amalgam; two small pieces of gold amalgam are to be seen adhering to the side of this stone. On one occasion when a somewhat large patch of antimony had been struck some tons of it were dressed to yield about 70 per cent. metallic antimony. No market, however, was found for it; either the antimony or the gold contents were negotiable, but not the two. The gold running to 35 oz. to the ton, the antimony was consequently sacrificed, and the stone was mixed in small quantities with the ordinary milling dirt and put through the battery, and the gold extracted in the ordinary way.

Conclusion.

The many interesting characteristics associated with the Lucknow goldfields tend to increase rather than to diminish the difficulty of writing a paper on the subject. Geological, mineralogical, mining, and milling descriptions have had to be given, and these in dealing with characteristics, many of which either have not been or have been but rarely met with elsewhere. There are points in connection with this field which claim the attention of each branch of the mining and metallurgical professions, and with so far-reaching an interest the writer cannot expect, nor does he hope, to escape criticism; such criticism can hardly fail to be instructive.

In dealing with the geological features the writer has endeavoured to give a full and clear description of each point that has been noted in the workings. In facing a problem which is undoubtedly a difficult one and capable of inviting different solutions, certain points

that to one may seem trivial to another may appear to supply circumstantial evidence towards the advancement of some particular theory. The all important question as to how the gold and auriferous pyrites became deposited in its present position raises the consideration of many points. What inference is to be drawn from the gold being found in shoots of calcite? Does the absence of gold in the quartz reefs at a distance from the joint suggest that a leaching process took place, which concentrated the finely distributed particles of gold into rich shoots at or near the line of contact; or does the presence of antimony and arsenic at once mark the deposits as being of deep-seated origin? Again, was the deposition of the gold contemporaneous with or subsequent to the earth rendings that resulted in the formation of the serpentine? Mention has been made of an important feature at the N.W. end of the ground, where the two diorites close in, only leaving a narrow band of soft "dig" intervening; how that in spite of new reefs being struck here, carrying all the other characteristics of the pay veins, no gold has been found in them. Also of certain veins, these too at the N.W. end of the ground, carrying gold downwards until the diorites close in and then become barren of gold. It would seem quite possible that serpentine will be found to widen out and again fill the space between the diorite and the veins recommence to carry gold, but nevertheless this failure of the pay shoots when serpentine is absent would appear to put beyond doubt the contention that the same forces that assisted the formation of the serpentine had much to do with the deposition of the gold. The final question, one of great interest alike to the geologist and to the investor, is, of course, what are the chances of continuation in depth, and this cannot be answered until a satisfactory solution to the above points has been arrived at, or the energetic prospecting now being carried on at the mines give actual proof one way or the other.

Looked at from a mining point of view it will be seen that unusual features have presented themselves, preventing the adoption of several accepted rules in working ground, or necessitating considerable modifications to them. These have not been dealt with in detail, because it has appeared to the author that conditions of an exactly similar nature are unlikely to be met with again, and that their record would render but little service. With such extensive workings (extensive if compared with the depth reached and with the small output of milling ore), and with men employed in faces remote from the shafts, everything is done to prevent time being lost. A general principle has been adopted. Thus, unless under

unusual circumstances, miners do not lay their own rails, fix on air pipes, put in timber, &c. Plate and pipe layers, timber men, and their assistants, are maintained, and they have to fly round, keeping the work up to date. In the case of the latter work timber hitches are marked out where necessary, and afternoon and night shifts cut them out ready for the timber men to work on during day shift. Nothing is allowed to hinder the actual work in the face. Contracts are let where possible, but the changes in the direction and in the class of rock to be traversed are so constant that the setting of piece work is difficult, and cannot be resorted to as frequently as desired.

Telephone communication has been used during the last twelve months, connecting the two ends of the mine and the manager's room. This has been of great service as a means of saving time and labour, and by allowing sudden changes in the country to be reported, so that unnecessary or ill directed work can be prevented. No accidents to life or limb have had to be reported during the past three and a half years, and though no comparison can be made with the working of dangerous mines, at the same time it may be useful, especially to the students of this Institute, to mention that this satisfactory record is attributed to strictness in punishing men for carelessness *without waiting for an accident.* A single example of a miner dismissed for working under baulked ground, instead of first taking it down, produces an excellent result, and the same may be said with regard to punishment for reckless handling of explosives, rushing of cages, &c.

The difficulty of preventing the stealing of rich ore is ever present, and not only is a large sum of money annually expended endeavouring to prevent loss by this means, but each move made in the mine and each appointment of foremen is influenced by this question. Precautions taken in one direction only appear to divert the danger into another. Changing rooms of course exist at each shaft, but nothing can prevent a certain amount of leakage through this channel. Collusion between several men was at one time very prevalent, but since the strike of 1897 this danger has been eradicated.

Mineralogists and milling men can hardly fail to find certain unusual and interesting features in connection with the auriferous arsenical pyrites and antimony.

In conclusion the author takes this opportunity of thanking the following gentlemen for their kind and able help rendered him in compiling this paper:—Messrs C. G. Warnford Lock, M.I.M.M. C. J. Alford, M.I.M.M., and H. Watson, F.I.C.

DISCUSSION.

The President thought they would agree with him that the paper was a most interesting one. It afforded scope for a good deal of discussion, more especially with regard to the origin of the reefs and as to the treatment of the ores. Mr. Alford, who was unable to be present, had asked him to read a short note on the subject of the rocks. Mr. Alford differed from the author as to the origin of the serpentine, and he thought it as well to read Mr. Alford's remarks before proceeding with the discussion on the paper.

Mr. C. J. Alford: In the few remarks which I have to make supplementary to Mr. Canning's most interesting paper on the Wentworth Proprietary Mine, N.S.W., I fear that I shall not have much sympathy from those mining men who look upon the introduction of technical geological problems into the arena of practical mining as superfluous if not even useless and misleading, yet to some I hope what I have to say may suggest questions of interest.

I visited the mine in October, 1896, and was accompanied through the workings by Mr. Canning. The workings were not at that time nearly so extensive as they are at present, as described by Mr. Canning in his paper, but that makes little difference in what I have to say about the mine. It struck me at the time that the supposition of an intrusive dyke of serpentine into the so-called "diorite," with a contact deposit of the auriferous mineral between the two could not be correct, but that all the geological conditions were pointing to a very marked example of transmutation or dynamic metamorphism. I found that the rock to the south-west was either massive or schistose, and that the joints were more or less filled with serpentinous material. Passing to the north-east the serpentine became much more plentiful, until the original rock was represented only by large detached masses or boulders imbedded in serpentine. Still further in the same direction these boulders became smaller, and the serpentine predominated, until at last the original rock passed altogether into serpentine with slicken-sided joints. In this zone, but more especially along the line of larger boulders, occur the veins and bunches of auriferous quartz and calcite, which have been described by Mr. Canning.

In Mr. Canning's section across the country from S.W. to N.E. the serpentine is shown in outcrops in the valley and the "diorite" on either side of it, overlaid by the auriferous alluvial deposit, which is covered on the surface by basalt. The lower rock, the

so-called "diorite," proves, under examination by the microscope, to be a magma basalt, or limburgite, composed of augite, magnetite, and pyrites, in a volcanic glass. Of the gravel and the upper basalt we have unfortunately no specimens before us, so we must be contented with their local denomination.

Now of this arrangement of rocks two explanations appear to be possible. One is that the whole is of one and the same rock in two flows, separated by a bed of tuffaceous material represented by the auriferous alluvial, and that the serpentine marks a zone of fracture throughout the mass with but little faulting or vertical movement. Another, and perhaps more probable, theory may be that the lower rock was fractured and thrown down to the N.E. and afterwards levelled on the surface by denudation. Upon this surface the alluvial bed and the upper basalt were afterwards deposited, and the valley subsequently cut out by erosion along the line of weakness caused by the original fracture below. It is in this line of movement that the alterations in the rock have occurred to which I desire to draw the attention of the meeting.

On the table are five rock specimens, numbered 1 to 5, from which the sections of the microscope slides, bearing the same numbers, have been cut. I have also, for convenience of examination, had the micro-photographs of the slides made, which are also before the meeting, and which I will now proceed to explain.

Slide No. 1 shows the magma basalt, or limburgite, as nearly unaltered as any specimen we have before us, a non-felspathic rock of basic composition, containing augite, magnetite, pyrites, and siliceous glass. This section shows the augite becoming ophitic round the edges of the crystals, where it is cut up by small veins and needles of some other mineral, but the separated parts, by showing optical continuity, prove themselves to be portions of the same crystal; this is the first setting in of decomposition. The pyrites and magnetite are opaque, and the glass is structureless, but easily recognisable by showing the same optical characteristics as the glass on which the section is mounted. This proves the rock not to be an andesite, as some persons have suggested.

No. 2 shows alteration proceeding, the augite passing into serpentine, and calcite developing in minute veins. Magnetite and pyrites are present, and also olivine as an original mineral, probably only accidentally absent from No. 1 section.

No. 3 shows serpentine with remnants of unaltered augite and pseudomorphs of serpentine after olivine; also the magnetite and pyrites increasing in quantity, probably derived from the decomposing augite.

No. 4 is a slice from the centre of one of the small boulders found embedded in the serpentine; it shows good crystals of augite in glass, and is identical with Section No. 1. The surface of this boulder will be seen to be serpentinised.

No. 5 is wholly serpentine with veins of calcite, but showing very interesting skeletons and outlines of the nearly obliterated crystals of augite.

We thus follow the transmutation of the augite rock into serpentine, and I will now pass on to consider very briefly some practical questions which this suggests regarding the future of this mine. We have seen in Mr. Canning's paper that the occurrence of gold is associated in this case in a large degree with that of calcite; let us see, therefore, whence this lime was derived. There are two suppositions which suggest themselves. There are in some parts of the adjacent country limestone hills which once may have formed parts of a very much more extensive deposit of that rock which has now been removed by denudation. It is possible that the calcite of the mine may have been derived from infiltrated water charged with lime from the decomposition of these rocks, and it is also possible that the gold may have been brought with it in the same manner from some surface deposits or disintegrating veins when the alluvial bed above the lower basalt was also enriched, and this I suppose will be the most generally received theory. In that case it is not probable that either the gold or the calcite will have been carried to any very great distance below the surface or beyond the range of percolating waters, and the prospects of the mine in depth will not be promising. There is, however, another possible theory which may be worth attention, and in order to illustrate this I will place before the meeting the following table, showing the constituents of the minerals concerned:—

	Augite.	Serpentine.	Olivine.
Silica	47·63	40·12	41 32
Alumina	6·74	2·00	0·28
Lime	20·87	—	—
Magnesia	12·09	40·04	54·70
Protoxide of iron	11·39	3·47	2·40
Protoxide of manganese	0·21	—	—
Water	0·28	13·36	0·30
	99·21	98·99	99·00

We notice here that the considerable proportion of lime contained in the augite does not pass into the serpentine. Might it not be that this has at least helped to form the calcite veins? And then, regarding the gold, there is in the basalt a certain amount of

magnetite and pyrites, and it is possible that gold may occur in or with these. If, then, the calcite and the gold were derived from the basalt their occurrence in depth will be probable. A very careful examination of the rocks would be necessary to settle these questions, and there are many others suggesting themselves.

Mr. J. H. Collins said that, with regard to the origin of the serpentine, he was, with the information before him, disposed to take Mr. Alford's view rather than that put forward by Mr. Canning, although he did not think they were in possession of sufficient details to enable them to form any very definite opinion on the subject.

He could not help thinking what an ideal place Lucknow would be for a skilled tributer to work in. If tributers could work to a large extent on their own account, and yet be helped in the larger general operations by the Company, it appeared to him that the men would do well, and the Company would probably do very much better than at present. Thieving seemed to be very prevalent, and the proportion of gold lost must be very great as compared with the amount the Company profited by.

In connection with the irregularity of the deposits which Mr. Canning had mentioned, the paradoxical remark heard in Mexico—"Never stop a drift until you have gone three feet further"—appeared to be exceedingly good advice. He thought if they wanted to make the best of such a country, they should get a man accustomed to watch the phenomena of deposits, and allow him, without much interference, to follow the faintest indication of ore; but the man with a theory was very likely to lose the rich deposits.

He was struck with the great cost of driving through the diorite, which, as far as he was aware, was the most costly rock to drive or sink through. He had known a case where the sinking of a shaft through diorite cost one guinea an inch, while £50 a fathom had been paid for a drift in Devonshire. It appeared to him that steel could not be made of sufficient hardness to bore through such rocks, and it seemed as if they would have to adopt some handy diamond drill for the purpose.

Mr. Butters considered it to be a very crude proceeding to put rock going about 250 to 300 oz. to the ton through a battery. It seemed to him that it would be much better treated in a small pan furnace and smelted. The antimony contained in the ore would roast off at a very low temperature without much loss of gold, and it could be mixed with litharge, or some flux, and smelted directly on the spot.

There was a good deal of material smelted in pots that was not as

rich as the ore in question. He would be glad to hear what was the value of the slimes, and what percentage there would be in the tailings. He also desired to know the number of tons of tailings on hand.

Mr. Canning stated that the slimes contained about 10 or 12 dwt., and two-thirds of the tailings were slimes. It was difficult to estimate the quantity of tailings, but there was probably an accumulation of about 60,000 to 70,000 tons of tailings, which averaged about 6 or 8 dwt. throughout.

Mr. Butters thought that under these circumstances it would pay to build cyanide works to treat the tailings.

Mr. McDermott said that, with regard to concentration, it was a little difficult to draw any conclusion as to the percentage of saving, but it seemed to him that the value of the concentrates was so great that a little closer attention to concentration would be worth while, and a double treatment of the pulp would doubtless lead to a still greater reduction in the value of the tailings.

Mr. James remarked that it had been tried at the Dolcoath without success.

Mr. McDermott stated that it was not correct to say double concentration was a failure, as in the case mentioned the extra saving was not sufficient to justify the extra installation; he thought that it depended upon whether it was worth while to adopt the double treatment, and in the present case he thought it was.

The occurrence of the gold was very interesting, but the bulk of the evidence appeared to show that the results obtained in the early days were much better than at the present time, and the bunches of rich ore were found more frequently. Mr. Alford had given them a theory which might account for that but, whatever might be the theory as to the occurrence of the gold, it seemed to be another instance of the tendency of the gold to concentrate near the surface, and not to get richer with depth. In the case before them the extremely high expenses in connection with the explorations, and the absence of anything but a mere contact line to follow as an indication, made it a most difficult proposition from a mining point of view for the investment of capital, and the local experience of men who had had a number of years' work there was of greater value than any number of theories.

Mr. Canning, in replying to the points raised, said that his remarks as to the genesis of the serpentine had been misunderstood by the President, in fact, his description and the samples exhibited, showed the passing of one class of stone into another, which was precisely Mr. Alford's theory.

The President pointed out that Mr. Alford spoke of the serpentine as a metamorphic rock, while the author referred to it as a dyke.

Mr. Canning remarked that the mistake had evidently arisen from the misapplication of the word "dyke." This word occurs, however, only once and then on the first page, where it is employed with the object of describing the appearance of the country; nothing is said later to endorse the "dyke" theory. Nevertheless, correctly speaking, it is clear that the use of the term dyke is misapplied and misleading.

Mr. Collins's suggestion as to tributing was an excellent one, and he hoped notice would be taken of it.

The question of theft was a very serious one, and actually caused a strike which lasted 3½ months. It appeared impossible to stamp the practice entirely out, and precautions taken in one direction only opened the door to theft in another. With such a small population it was difficult to employ detectives, as the whole of the people would be watching them, instead of their being able to watch the people unobserved.

With regard to the hardness of the rock, they sent down 18,000 sharp drills every week, which was a very large number. A diamond drill as suggested by Mr. Collins was used as much as possible, and on soft rock did excellent work—sometimes 12 to 15 ft. a shift; through the hard rock, however, they could not do more than 4 or 5 in. per shift.

With regard to Mr. Butters's remarks as to the rich stone, there was little or no loss in treating it. They obtained every bit of free gold on the plates, and the remainder was a very simple process —it was just run into settling tanks, and there was practically no loss whatever. The slimes running away from the settling tanks had been assayed, and although the result was rather high the quantity was small, and it passed into the ordinary tailings heap to be recovered in the usual way.

Mr. McDermott recommended the paying of closer attention to concentrating, namely by the double treatment of the pulp. It may be noted that trials have been given to this as well as to other methods. By using more vanners, thus reducing the burden on them and increasing their efficiency, the temporary loss in the tailings had been reduced very considerably, but not to the point that it was hoped would be reached by cyaniding. Again, the further outlay involved and maintainance of these extra vanners had been very closely gone into, and it was clearly shown that the adoption of this suggestion was not advisable. Re-vanning the tailings d

not recommend itself for the same reason. Hydraulic separators, placed between the copper-plate tables and the vanners, were also given a trial but were discarded. Regarding the future treatment of the tailings, he thought it would be expedient to adopt filter presses at Lucknow. He had seen them at work in Western Australia on tailings carrying large quantities of slimes, and their introduction had been attended by success in every way.

"Notes on the Electro-deposition of Palladium."

By SHERARD COWPER-COLES, M.Inst.M.M.

PALLADIUM is usually found associated with platinum, and as a rule it does not exceed 1 per cent. It is generally separated from platinum by chemical processes.

At the present time the uses for palladium are very limited, the demand being considerably less than the supply, consequently the price is low as compared with some other rare metals. The present price is about double that of platinum, but its specific gravity is about half. Its colour is almost as white as that of silver, and it takes a beautiful polish. Palladium is very malleable and ductile, and is harder than platinum. The following table gives the hardness of electro-deposited palladium as compared to other metals:—

Nickel electro-deposited	10·0
Sheffield plate	10·0
Antimony electro-deposited	9·0
Palladium deposited bright	8·0
Cadmium deposited bright	4·5
Silver electro-deposited and burnished	4·0

The figures give the hardness as represented by the number of grams weight on a diamond point required to produce a scratch.

Palladium fuses at a somewhat higher temperature than platinum. Moissan has found that palladium, like platinum, dissolves carbon at the temperature of the electric furnace, but does not unite with it, the whole of the carbon separating out as graphite on cooling.

The author has recently used palladium as a protective coating and reflecting surface for parabolic reflectors, some specimens of

which are on the table, the whole of the parabolic reflector being made by electro-deposition. The palladium can be deposited quite bright on the surface of the mirror, so that it does not need polishing, which would destroy the true parabolic curve. A deposit of 70 to 80 grains per superficial foot is found to afford a good protective coating. Such a film of palladium is probably transparent, in which case a considerable portion of light would be reflected from the silver surface beneath the protective film of palladium.

Palladium is unchanged by air at ordinary temperatures, but at a moderately high temperature a thin film of oxide is formed, which has a bluish colour. At a higher temperature the oxide is decomposed. Palladium alloyed with one-tenth silver is sometimes employed for the graduated scales of astronomical instruments.

Smee devoted some attention to the electro-deposition of palladium, and attempted to deposit metallic palladium from the nitrate, but only succeeded in getting down a black powder. He found ammonio-muriate of palladium dissolved in ammonia to be the best. He also tried iodide of palladium, by adding a solution of iodide of potassium to a solution of palladium, but found it was not a suitable salt for electro-deposition. He also tried palladio-cyanide of potassium. An electrolyte made up of cyanide of potassium is said to take up a considerable quantity of metal and yield the metal freely, so that heavy, reguline deposits can be obtained.

Bertrand has succeeded in obtaining good deposits with a solution formed of the double chloride of palladium and ammonium, made perfectly neutral, a palladium anode was found to dissolve freely.

Gore has electrolysed a solution of chloride of palladium, in which a palladium anode was found to dissolve, and a black deposit of palladium was obtained upon the cathode.

The author has found a solution of palladium ammonium chloride to be one of the best, if not the best, for depositing palladium in a bright form, such as exhibited here to-night. The best proportions are about a 0·62 per cent. solution of palladium ammonium chloride dissolved in about a 1 per cent. solution of ammonium chloride. The solution is worked at a temperature of about 75° F., the current used per sq. ft. being about 0·12 ampere, the E.M.F. at the terminals of the bath being 4 to 5 volts. The anodes can be either palladium foil or carbon.

DISCUSSION.

Professor Huntington said that Mr. Cowper-Coles appeared to have found a very useful application for palladium. He used it for mirrors, which apparently were made of copper, silvered, and then coated with palladium; but he did not say for what purpose it was used. He (the speaker) understood that it was to protect the reflectors from the action of the salt contained in the spray; most other metals being liable to be attacked by the salt. The quantity of palladium available in the world was very small, so that its application was limited.

The author made a point in his paper that a considerable portion of light was reflected from the silver surface beneath the palladium. He did not see that it was a matter of any great consequence whether the light was reflected from the silver beneath or from the palladium surface, neither did it appear to be very probable.

Mr. Sayers, on behalf of the author, said that Mr. Cowper-Coles had found palladium to be better than any other metal for resisting the action of salt. The reflectors, coated with palladium, were used at sea in search-lights. With regard to the transparence of the palladium, it was merely a supposition on the part of the author, and he did not think it was of any particular importance.

The President said, that owing to the lateness of the hour it was not possible to read Mr. Kendall's paper, which was of a very voluminous character, and deserved careful study. Under the circumstances he proposed to take Mr. Kendall's paper as read, and invite any discussion at next meeting.

"The Silver-Lead Deposits of the Slocan, British Columbia."

By J. D. KENDALL, M.Inst.M.M.

INTRODUCTION.

BRITISH Columbia will in all probability in the near future become one of the most important producers of silver-lead ore in the world, if indeed it will not occupy the first position. It is not that the deposits are large; rather they may be said to be numerous and the ore high grade. The district known as the Slocan contains most of these mines hitherto opened up in the province, and therefore gives a fair idea of B.C. silver-lead mines.

The first mineral locations were made, in the Slocan, in September, 1891; so that as a mining district it is in its infancy. When in conjunction with this we remember the great difficulties in transportation that had to be overcome by the pioneers of the mining industry, we are surprised that the output of ore is so great rather than it is not greater.

A few of the earliest claims of importance appearing on the records may here be named. The first is the Payne, located on the 9th September, 1891. Others were located soon after, as shown by the dates of record below.

Name of claim.	Recorded.
Payne	22nd September, 1891.
Noble Five	5th October, 1891.
Last Chance	5th October, 1891.
Slocan Boy	5th October, 1891.
Slocan Star	15th October, 1891.
Washington	17th October, 1891.
Idaho	4th July, 1892.
Ruth	8th July, 1892.
Alamo	5th August, 1892.

On each of these claims shipping mines have been opened up, and most of them are to-day on the list of shippers.

The first shipment of Slocan ore was from the Whitewater Mine in July, 1892. It had to be "packed" on horseback through the forest, a distance of 17 miles, to Kaslo, at a cost of about $40 per ton. Then it was taken by boat and rail to United States smelters at a further cost of about $20 per ton. Ore was packed in a similar way from the Best and Dardanelles a few weeks later, and from the Idaho and Mountain Chief Mines during the winter of 1892–93. The cost from the two last-named mines was $45 per ton to Kaslo, and from other mines at somewhat similar rates.

How near the surface the ore occurred and with what little preliminary work it was reached, outside the difficulties of transportation, may be judged from the fact that a quantity of ore was shipped in 1892, in most cases, however, only a sample. The following figures are from the Report of the Minister of Mines for 1892:—

Name of mine.	Ore shipped. Tons.
Whitewater	8
Freddy Lee	over 400
Blue Bird	100
Dardanelles	10
Best	15
Idaho	15
Wellington	10
Noble Five	not stated

The first railway built into the Slocan was the Nakusp and Slocan, commenced July, 1893, and opened for traffic as far as Three Forks (32·9 miles) in 1894. This was followed, in 1895, by the Kaslo and Slocan (31·8 miles long). In the same year the Nakusp and Slocan was extended from Three Forks to Sandon, a distance of about 4 miles. These lines placed the Slocan in communication with the Canadian Pacific, Northern Pacific, and Great Northern railway systems and reduced the cost of transportation $20 per ton or thereabouts.

Location and Extent of District.—The Slocan mining district is in West Kootenay, on the eastern side of Slocan Lake. It is about 12 miles across in an east and west direction and about 18 miles long from north to south. Part of it, on the east, is really in the Ainsworth mining division, but there is no natural line separating the Ainsworth from the Slocan mining division. It is merely an arbitrary line for record purposes.

Physical Geography and Geology.—The entire area is mountainous, ranging in altitude from about 1,800 to about 7,500 ft. The

mountains are not only high but steep. Usually they are very uniform in outline, a result of the slight variation in the hardness of the rock out of which they have been formed, over a great part of the area.

The solid geology is relatively simple, the greater part of the area being occupied by dark argillites or clay slates occasionally interbedded with calcareous quartzite and dark coloured limestone, see Fig. 2. Traversing these rocks are numerous igneous dykes and intrusive sheets, some of great thickness. Many of them are closely associated with the ore deposits as will be seen hereafter. A number of these igneous intrusions may be seen in the neighbourhood of Three Forks along the line of the Canadian Pacific Railway. In a cutting on that railway, just below Three Forks, the section shown in Fig. 2 is exposed, and about half a mile west, on the way to the Alamo Concentrator, commonly called "The Concentrator," a dyke, about 150 yards wide, crosses the railway. Many others may be seen in the same locality and elsewhere in the district.

FIG. 2.—Section in railway cutting at Three Forks.

A, argillite; B, felspar porphyry.
Scale 50 ft. to an inch.

The aqueous rocks are known to geologists as the Slocan slates. They seem to occupy about the same geologic horizon as the upper part of Adam's Lake Series, which has been provisionally classed by the Dominion Geological Survey as Silurian or Cambro-silurian. The beds are, as a rule, highly tilted and severely faulted. Most of the faults are ordinary, but some are reversed. Both kinds are met with in the mines and some of them frequently stagger the so-called practical miner who, unfortunately, is very often only a practical rock breaker and absolutely innocent of any knowledge of geological structure.

The remainder of the area is occupied by granites and granitoid rocks.

Climate and Forests.—The climate of the Slocan is very good. There is, however, an almost entire lack of precise information regarding it. Generally it may be said that the snow "comes to stay" on the higher altitudes in October, and throughout the district in November. It disappears from the lower levels at the end of March or beginning of April and from the mountain tops, except

where drifted, about the end of June. A little rain falls in the autumn and spring, but the summers are long, dry, and bright. The following table shows the depth of snowfall at the Idaho Mines (altitude 6,100 ft.) during the winter of 1897–98:—

		Snowfall. Inches.	Days on which snow fell.
1897.	September	3	1
	October	12	2
	November	95	22
	December	56	13
1898.	January	55·5	15
	February	104·5	19
	March	44	8
	April	25·5	9
	May	24	7
		419·5	96

At the Corinth Mine, not more than a mile from the Idaho, and at an altitude of about 6,200 ft., the following figures were recorded during the same winter. First snow 12th October, 1897. Between that date and the 22nd of April, 1898, 394 in. of snow fell. The record was not regularly kept afterwards, but on the 14th and 22nd of May 2 and 3 in. respectively fell.

No exact figures can be given with regard to rainfall or temperature, but with regard to the latter it may be stated that unmelted ice is frequently seen in July and August in the lowest or drainage levels of mines near the mountain tops. It may also be stated in a general way that in summer the thermometer sometimes rises to 90° in the shade, with cool nights, whilst in winter the mercury may often be seen in the neighbourhood of zero, and occasionally 20° or more below.

Before the advent of the prospector the Slocan was thickly covered with firs and pines, except on the higher mountain tops. Since then large areas have been partly or wholly burned, but many of the trees within those areas, although killed by fire, still stand and are quite good for mining purposes. The firs are more useful to the miner than the pine. The principal species of the former are:—

Pseudotsuga Douglasii	Douglas fir or red fir.
Tsuga mertensiana	Hemlock.
Thuya gigantea	Cedar.
Larix occidentalis	Tamarack.
Abies subalpina	Balsam.

The balsam ranges in altitude from about 3,000 to about 7,000 ft., the cedar up to about 5,500, the hemlock and Douglas fir to about 3,500 and 4,000 feet respectively.

Transportation.—The mines are connected with one or other of the railway systems serving the district by either rawhide trails, wagon roads, aërial or other tramways.

Rawhiding is extensively practised in the Slocan, the conditions being highly favourable for it, the steep mountain sides and the thick mantle of snow by which the ground is covered throughout the long winter making it possible to form a trail for use in this way at a cost little exceeding that necessary to remove the underbrush and the few large trees that cannot be avoided along the selected route. In rawhiding, the ore is first sacked—each sack containing 100 to 150 lb. of ore according to the tenor reached in dressing, then wrapt in raw cowhides—15 sacks being an ordinary load—and drawn by horses over the snow. A horse takes only one hide at a time, but one driver may be in charge of any number of horses up to four, the number depending upon the character of the trail. This system of haulage can only be followed in winter when a good covering of snow is on the ground.

In some cases wagon roads are built to the mines. Then the haulage is done by sleighs in winter, and by four-wheeled wagons in summer, the load (depending mainly upon the grade) ranging from 4 to 8 tons. These roads cannot be used for about four or five weeks in the spring—when the snow is melting—because part of them are then only fit for "runners," and part for wheels. As the melting snow disappears at the foot of the hills, runners are impossible, and wheels cannot be used until the snow has gone entirely from the higher ends of the roads. The usual time that elapses after the ground is bare in the lower valleys, until the snow disappears at an altitude of about 6,000 ft. is four or five weeks, and during this time haulage by this system has to be suspended, to the higher mines.

Some of the mines are provided with rail, others with rope, tramways—in all cases self-acting. Both these forms of tramway, if properly arranged, are capable of continuous working throughout the year, and are therefore superior to either rawhiding, wagon, or sleigh haulage, where the output is large enough to justify the extra cost of making them.

The longest wagon road yet made is about 8 miles, the longest tramway about 1¼ miles.

Form of the Ore Deposits.—Most of the deposits hitherto worked are at considerable altitudes, for the simple reason that the rocks are more exposed there than at lower levels, and consequently the prospector has been more successful. The following table shows approximately the level at which some of the more important

mines occur, disregarding the range of altitude in the several properties :—

Mine.	Feet above sea.
Payne	7,100
Idaho	6,700
Alamo	6,700
Reco	6,200
Whitewater	5,000
Slocan Star	5,000
Ruth	4,700

Nearly all the deposits known in the Slocan, and certainly those of greatest importance commercially, have the form of veins, often called, here and elsewhere, "fissure veins" or "true veins," but a few are bed-like in form, that is, occur along the bed-planes of the strata. The two kinds will be considered separately.

Veins.—Most of the deposits in this class, hitherto worked, occur in the argillites or clay-slates, a few only having been worked in the granite area. Whether this preponderance is due to the former rock being more favourable to the formation of ore deposits, or to the greater ease with which they can be opened up after they are found, is not evident. Experience on the granite area is too small for any reliable judgment to be pronounced yet, but the development of the Enterprise Mine on Ten Mile Creek, has shown conclusively that it is possible for good mines to occur in that area.

The *direction* or *strike* and the *hade* or *underlie* of some of the veins is shown in Fig. 3, which gives the information in a form that admits of ready reference and comparison. One fact is rendered very evident by this drawing—there is no particular direction for these veins, as is often assumed, although it must be admitted they seem more frequently to have an easterly and westerly bearing or strike than any other. The statement sometimes made that what is called a "true fissure vein" must have an inclination or underlie of at least 60°, is also seen to be erroneous, for all the veins included in Fig. 3 are of that class, yet some of them occur at lower angles than 45° even. As in other areas, the veins here, considered singly, are not constant either in direction or hade; the lines in Fig. 3 represent only the average direction, so far as at present known. The changes in direction are fairly well illustrated by the horizontal section of the Payne vein in Fig. 4, and the variations that occur in hade are suggested by the cross-section of the Alamo vein in Fig. 5. Some of the variations in the different veins are more precisely indicated on Fig. 3, the numbers thereon showing the inclination in degrees, of the respective veins, from a horizontal plane. Where two sets of figures are given, they indicate the range of variation in

hade. All the veins named in Fig. 3, except the Comstock and Enterprise, are in the argillites and their associated rocks. The

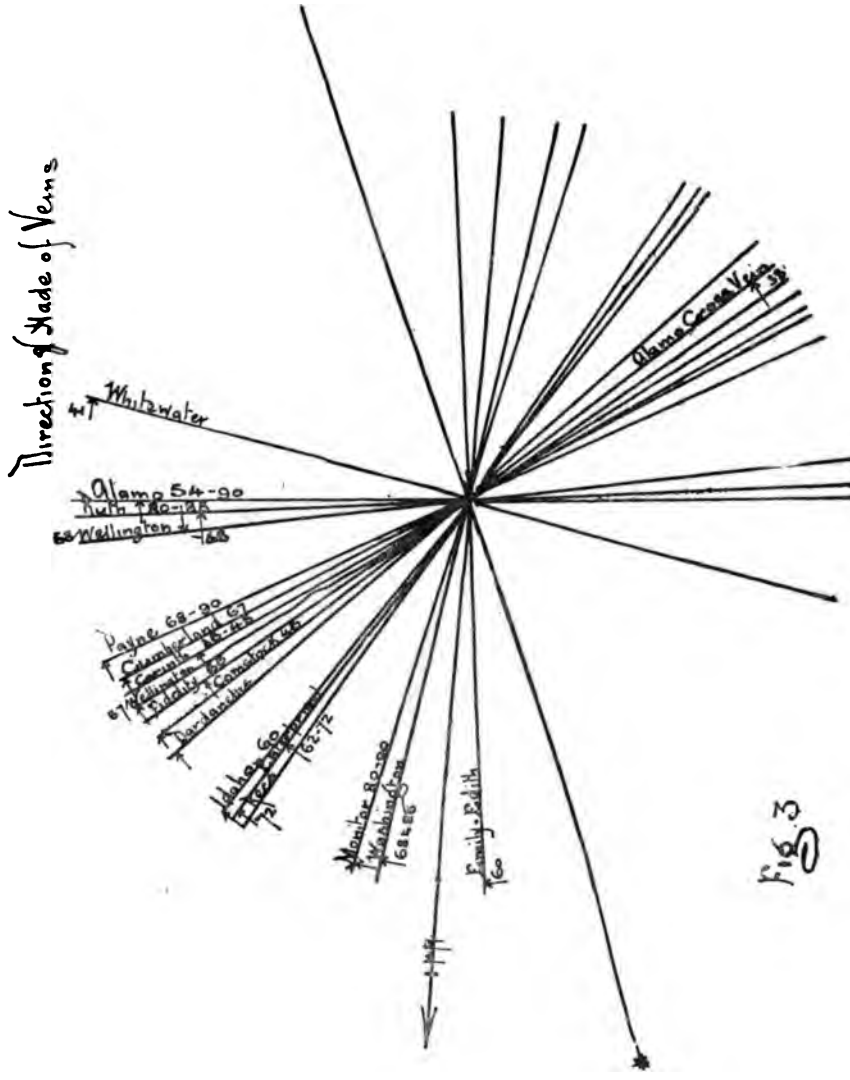


Fig. 3

two last mentioned veins are in mica-syenite and granite respectively.

The greatest proved *length* of any one vein in the argillites is only a little over a quarter of a mile, and only about a third of a

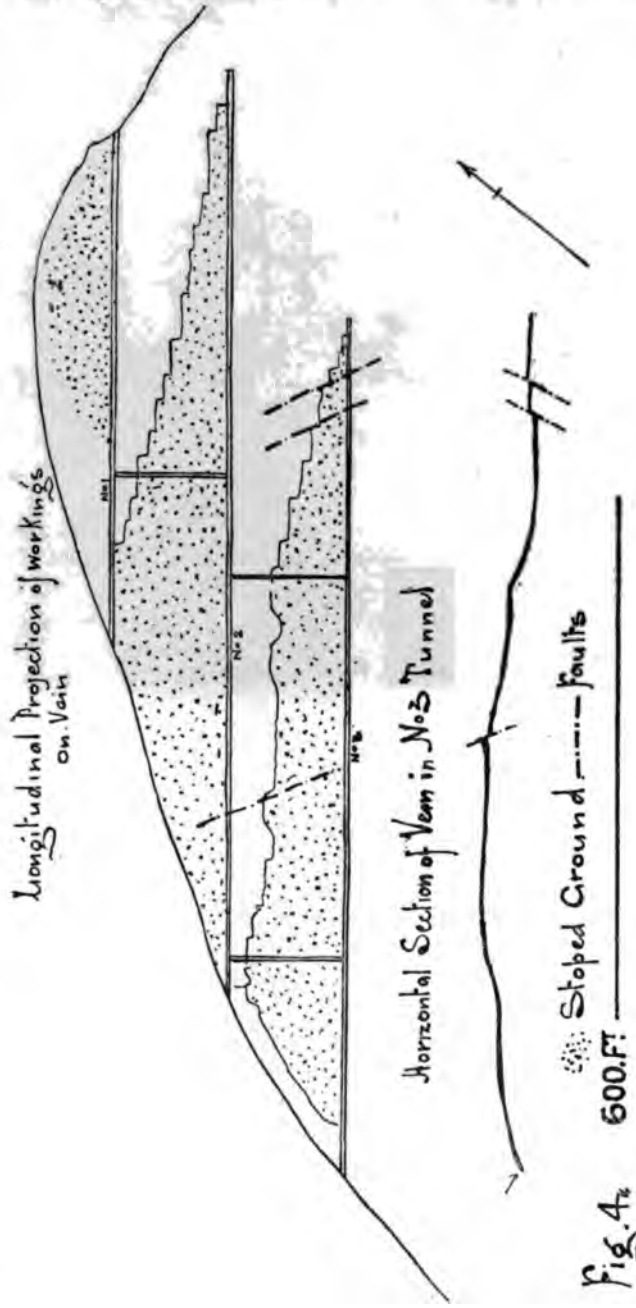

Longitudinal Projection of Workings on Vein
No. 1
No. 2
No. 3
Horizontal Section of Vein in No. 3 Tunnel
Stoped Ground
Faults
600 FT.

Fig. 4.

mile in the granitoid rocks. This statement will doubtless be disputed by those whose powerful imagination can supply the facts which observation fails to find, but it is true nevertheless. How

FIG. 5.—Cross-section of a vein. (Alamo Mine.)

Scale 200 ft. to an inch.

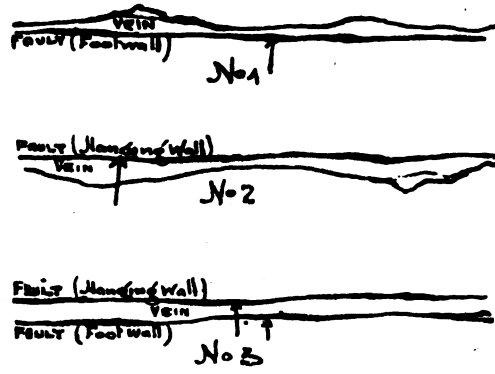
much longer they may hereafter prove to be we cannot tell. What has been said, however, may be taken as a protest against the too common practice of *assuming* the existence of veins in areas that are covered by drift or otherwise, and in consequence of which the solid rocks cannot be seen. Inferences as to continuity in such cases can only be drawn with safety when the observed facts, on opposite sides of the concealed area, are in complete accordance.

The *width* of veins is likewise very variable. In the argillites the same vein may, in parts, be only a few inches wide, whilst in other parts it may be over 30 ft. The veins in the granite are less variable than those in the argillites, and are seldom wider than 48 in., although, like those in the latter rocks, they occur of all widths below the maximum, the tendency being, in both classes of rock, for the veins to form a connected succession of truncated lenses.

In some case veins, or parts of veins, occur on one side or other of a dislocation or fault, the plane of which forms either the hanging—or the foot wall of the vein, according to the side of the ore on which the fault exists. In such cases, the side of the vein adjoining the fault is fairly regular, and the variations in the width of the vein arise from irregularities in the opposite wall, as shown in drawings I and II, Fig. 6. Or a vein may occur in part, or wholly, between two faults, as in drawing III, Fig. 6, the planes of the faults, in such cases, forming the hanging wall and the foot wall

respectively. The width of the vein is then much more regular than when the ore follows the line of a single fault, as in I and II, Fig. 6.

FIG. 6.

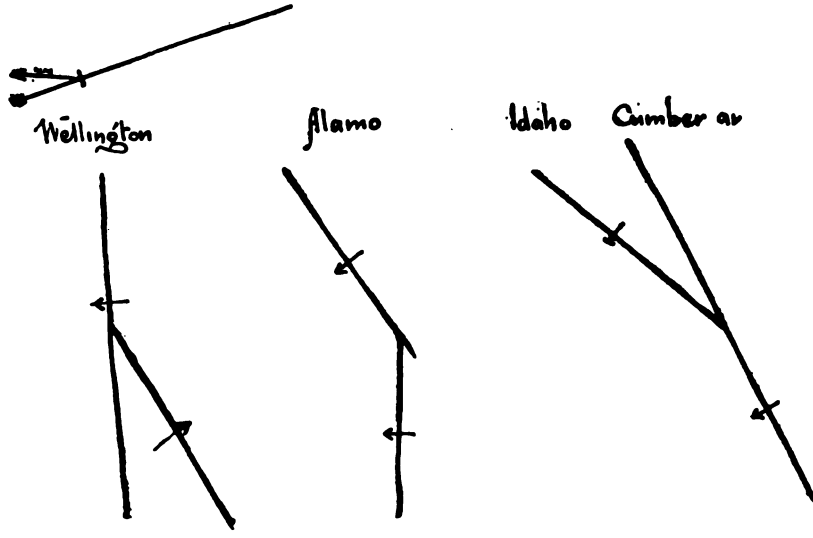


The greatest *depth* that has been reached on any vein is about 450 feet, but the bottom of the ore-chute had not been reached, so that the question of depth is entirely unsolved.

Veins usually occur singly, but in a few cases two, having different directions, are found together, crossing one another occasionally, and most probably always, but this cannot in every case be tested by observation, as one of the veins (which apparently cross) is often, if not always, dislocated and shifted by the other. Three instances of this kind are given in Fig. 7. It has just been stated that some veins, or parts of veins, occur along the line of single faults. Other veins, or other parts of the same vein, occur between two faults, and it may now be affirmed that, perhaps, every vein is on the line of one or more faults or dislocations of the strata. This is clearly shown by the severe abrasion of the walls so frequently observable, the broken-up condition of the country rock occurring between the walls, which usually forms such a large proportion of the gangue in every vein, and by the different dip of the strata on opposite sides of the vein. In some cases there is only one fault, in others two or more occur quite close together. The movement along these various fault-planes has often been in quite different directions. It is not unusual, for example, to find the striæ on one of these planes running thus //////, whilst a few inches behind, on a parallel plane, they have a direction at right angles so, \\\\\\. These striæ are of great significance in the dynamics of faulting. In some places they are horizontal, or nearly so. In others they have the same hade as the fault plane on which they

occur, whilst occasionally, as just stated, they have intermediate directions. The abrasion has in some instances—where the nature of

FIG. 7.



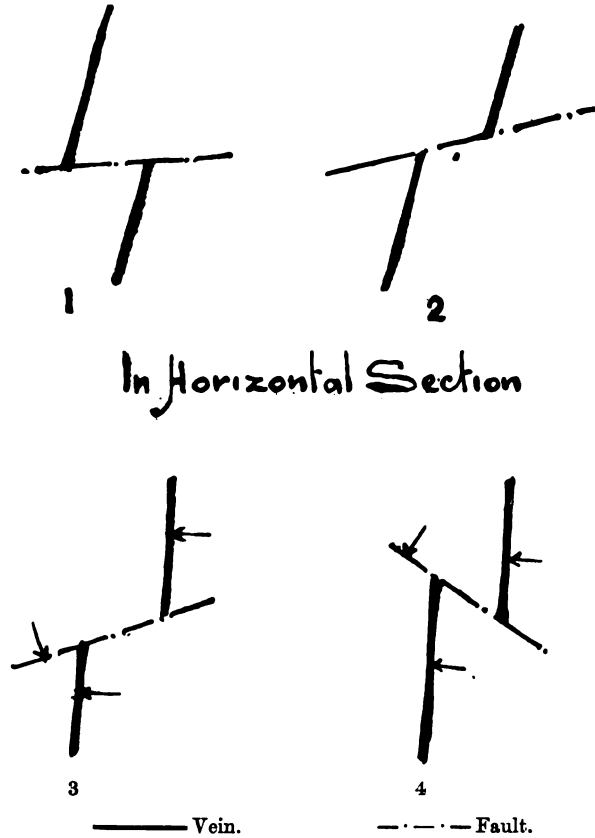
the rock was suitable—produced the highly polished surfaces known as "Slickensides."

In addition to the faults just named, every vein, perhaps, has been intersected crosswise by one or more faults. These may be divided generally into two groups, one having a direction more or less parallel to the vein, the other group more or less at right angles to it. Examples of the former are given, in vertical section, and of the latter in horizontal section, by Fig. 8. Nos. 2 and 3 are ordinary faults, Nos. 1 and 4 reversed faults. In each of these cases, except No. 1, Schmidt's rule, for finding the shifted part of a vein, holds good. No. 1 is quite an exceptional kind of fault, only two or three examples having come under the writer's notice in the Slocan. When confronted by a fault of this description, it is necessary to be on the lookout for any indication of drag, such as is shown at A in Fig. 9, although that is an ordinary fault; or it may be possible to obtain a clue to the shift like that given in Fig. 10, which is a cross-section of a drift in the Ruth Mine, showing two of these reversed faults. The section, however, indicates the true nature of the lower one only. Fig. 11 shows two of these reversed faults in longitudinal projection. The effect of any of these faults is to shift the veins they intersect, either to one side or the other, and for variable distances. A case in

which the shift is large is shown in Fig. 12, which is a plan of No. 3 tunnel in the Enterprise Mine. The vein, as will be seen from the

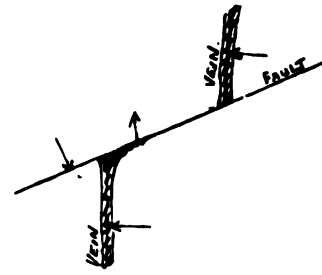
FIG. 8.—Effects of cross-faulting.

In vertical section.



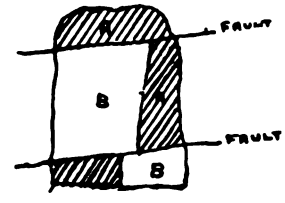
drawing, is intersected by four cross faults close together. The net amount of shift or heave is about 100 ft. In other mines, like the Payne for example, the shift of the vein by these cross faults is very little, as may be seen on reference to Fig. 4. The *amount* of shift on any vein is very variable. The *direction* of shift, on the contrary, is very frequently, in fact generally, the same, on the same vein; but there are exceptions, as in the Payne Mine for example. Some of the faults which cross that vein shift it to the north-west, others to the south-east, as may be seen on reference to the horizontal section

FIG. 9.—Dislocated vein. (Enterprise Mine.)



Scale 8 ft. to an inch.

FIG. 10.—Reversed faults as seen in tunnel face. (Ruth Mine.)



A, argillite; B, vein.
Scale 8 feet to an inch.

in Fig. 4. An instance of a vein being frequently shifted in the same direction is given in Fig. 13.

In the neighbourhood of most veins occurring in the argillites, dykes and intrusive sheets of felspar porphyry are found. These, in some cases, cross the line of the vein; in others they are approximately parallel to it. A case of the latter kind is given in Fig. 14, which is a cross-section of the Slocan Boy vein. The workings in that mine have been carried to a depth of 200 ft., and, so far as they have yet gone, the porphyry either forms the hanging wall of the vein, or is separated from it by only a few feet of argillite. When the dykes or intrusive sheets cross the course of a vein they almost invariably furnish additional evidence of the faulting that has taken place along the line of the vein, for they seldom, if ever, intersect the vein but are intersected by it, and shifted as shown in Fig. 15. Further evidence that the faults coinciding with the veins have had one or more movements since the intrusion of the igneous dykes and sheets is supplied by Figs. 16, 17, and 18, fragments of igneous rocks having, in each case, been torn from the main body and dragged through the crumpled, crushed, and kneaded argillites.

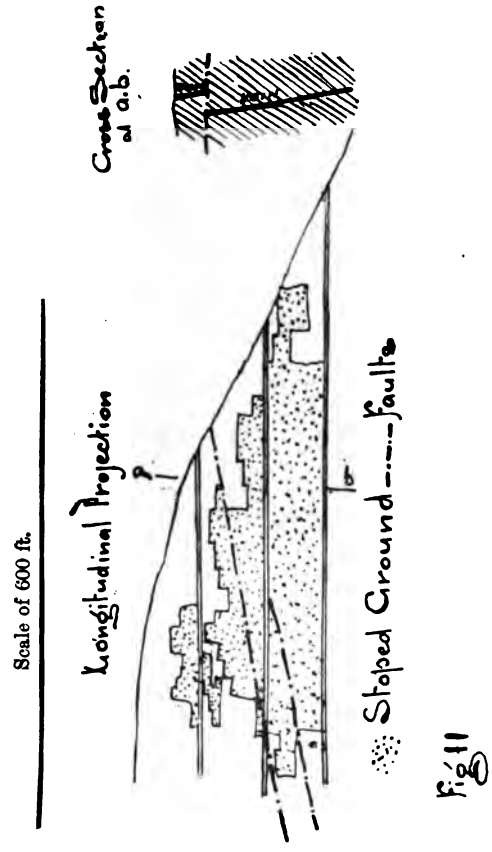
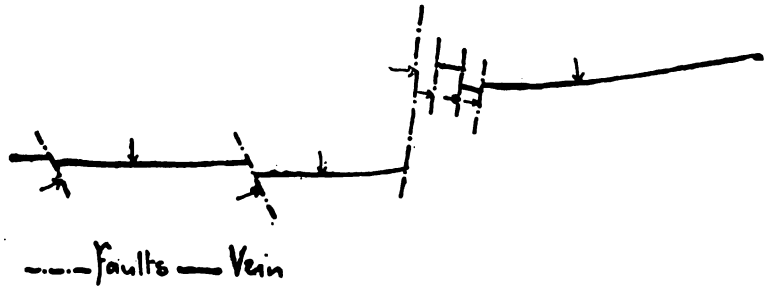


FIG. 12.—Horizontal section of vein in No. 3 Tunnel. (Enterprise Mine.)



Scale 200 ft. to an inch.

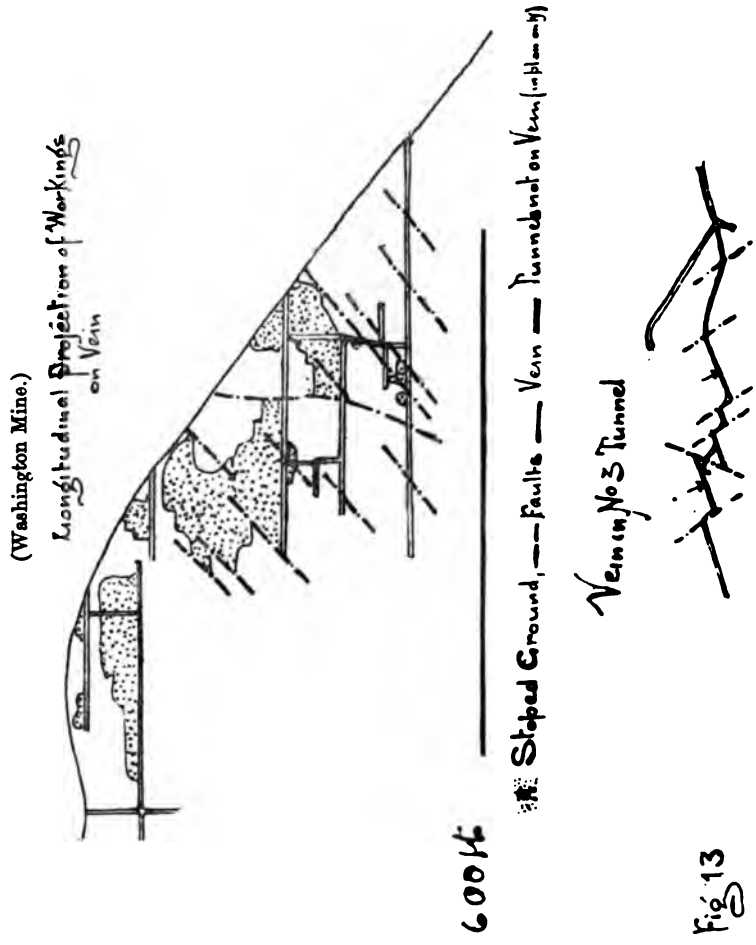


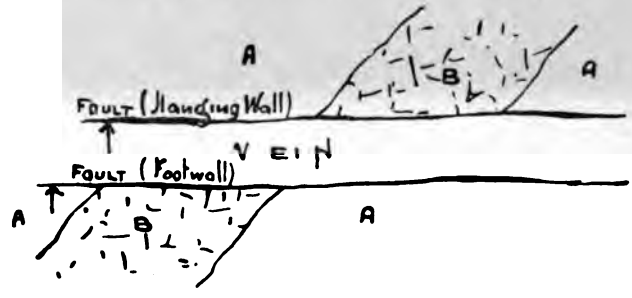
Fig. 13

Bed-like Deposits.—Examples of this kind of deposit are few, and they have not been worked to any large extent. Fig. 19 reproduces a section exhibited near the upper end of Spring Creek. The ore occurs in lenses along the bed-joints of the limestone. When the deposit, seen in the section, is followed lengthwise in the tunnel, it nips out entirely. A few yards of barren ground occur, and then a lens, somewhat like the last, comes in, but on a different bed-joint. A third lot of ore, practically like the other two, is on a different bed-joint again. The direction of the deposit (or deposits) is N. 76 E. and S. 76 W., and it hades southwards, at angles ranging from 75° to 88°. Cross-cuts have been made in the tunnel to both contacts, but ore was not found there.

FIG 14.—Vertical section of vein at mouth of upper tunnel. (Slocan Boy Mine.)

A, argillite, thin bedded and crumpled; A*a*, thicker bedded; B, felspar porphyry.
Scale 5 ft. to an inch.

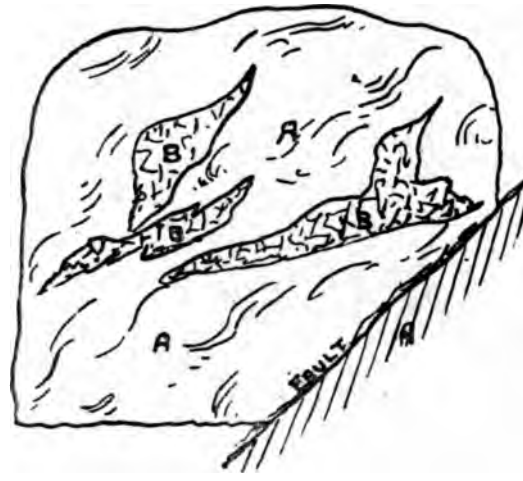
FIG. 15.—Horizontal section. (Alamo Mine.)



A, argillite; B, felspar porphyry.
Scale 10 ft. to an inch.

Another deposit of this nature has been worked in a small way at the Great Western Mine. Fig. 20 shows a part of that deposit in horizontal and vertical section. The ore, as will be seen from those drawings, occurs on the contact between argillite and an intrusive sheet of felspar porphyry. In one place it was found extending across the porphyry, between the two contact deposits, as shown in the horizontal section, Fig. 20, but in other parts (in either deposit) it does not leave the contact, towards either the porphyry or the argillites, more than 20 in., so far as exposed by the present workings. The direction of the deposit is N. 20 W. and S. 20 E., and it hades eastwards at an angle of about 50°.

FIG. 16.—Section in tunnel No. 3, showing occurrence of irregular fragments of eruptive rock in vein. (Alamo Mine.)



A, argillite, crushed and crumpled, and bedding partly obliterated; B, felspar porphyry.
Scale 4 ft. to an inch.

FIG. 17.—Irregular fragments of eruptive rock in vein. (Alamo Mine.)

A, argillite; B, argillite, crushed, crumpled, softened, and bedding in great part obliterated; C, felspar porphyry; D, galena.
Scale 4 ft. to an inch.

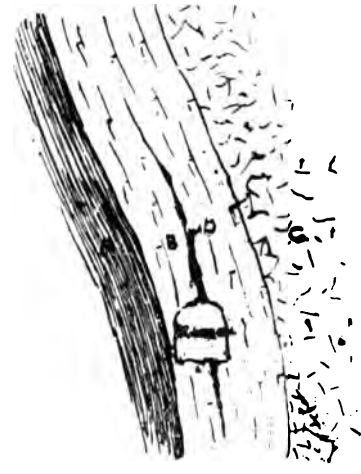
Deposits of this kind may be intersected and shifted by faults exactly like veins, an instance, in support of this statement, being given in Fig. 20.

Inner Nature of the Deposits.—What is commonly known as "vein matter," that is, the non-metallic minerals which make up the whole or a large part of every vein, consists, in great part, of more or less

FIG. 18.—Vertical section across disturbed part of vein. (Alamo Mine.)

A, argillite, crushed and crumpled, and bedding partly obliterated; B, felspar porphyry; C, galena and blende.
Scale 8 ft. to an inch.

FIG. 19.—Vertical section. (Carbonate No. 1.)

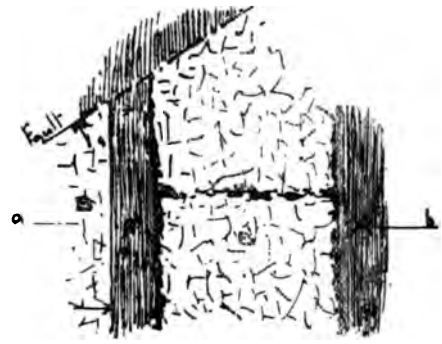


A, argillite; B, limestone; C, granite; D, galena and limonite.
Scale 20 ft. to an inch.

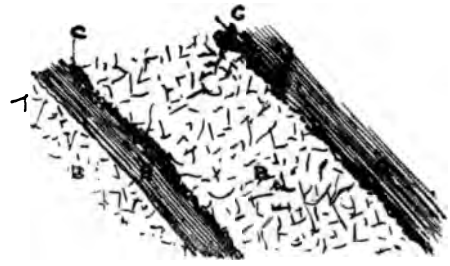
altered country rock with varying proportions of quartz, occasionally some calcite or dolomite, and very rarely felspar. If the country rock be either argillite or granite, or in fact a siliceous rock of any kind, quartz is the predominant xenogenous non-metallic mineral in the vein, calcite appearing then but rarely. On the other hand, if the country rock be calcareous, very little quartz is

found among the gangue, but a large proportion of calcareous matter. The alteration of the country rock included in a vein may be, and often is, of two kinds—mechanical and chemical. The mechanical alteration has mainly resulted from abrasion, the

FIG. 20. (Great Western Mine.)



Horizontal section.



Vertical section on *ab*.

A, argillite; B, felspar porphyry; C, galena, blende, and quartz.
Scale 20 ft. to an inch.

hardened rock having been ground into powder, and, where sufficiently argillaceous, kneaded, with the help of water, into "gutta-percha" or soft putty-clay. In some cases the mineralogical constitution of the country rock is such that any amount of grinding and kneading would not produce either of these clays. The abraded material then appears in a less coherent form and more like a fine clayey sand. In either case the pulverised rock usually adjoins one or both walls, if there be two lines of faulting, and is known to the miners here as "gouge," elsewhere as selvage or flucan.

The chemical alteration induced in those portions of the country rock which occur within the vein (and often for some distance into the walls) consists in the partial removal of the alkalis and more

basic constituents. The course of some of these chemical alterations may often be observed, as, for example, in the Comstock Mine on Fennell Creek. There the country rock is a mica-syenite, but, in those portions of it included in the vein, mica is usually absent. Here and there it is possible to find the faintest trace of the crystals of that mineral, the iron having been removed. In such cases if the rock be carefully examined inch by inch towards the unaltered walls, it will be seen that the flakes of mica become gradually more and more distinct, until, in the unaltered rock, they assume their normal appearance.

The quartz which makes up a considerable part of the Slocan veins has often a tendency to occur in bands, more or less parallel to the walls, of varying and irregular widths and frequently interrupted by the country rock included in the vein. Strings and bunches, along lines of jointing, are also of common occurrence, the ramifying quartz in this way sometimes producing a most complicated network in the rock which forms the greater part of the remaining gangue in the veins.

In addition to the vein matter just described, all veins perhaps contain a larger or smaller proportion of the metallic minerals. Those met with most frequently are galena, blende, and siderite, the first and last being often altered, especially near the surface, into cerussite and limonite respectively. In smaller quantities the following are met with:—(1) Pyrite, (2) tetrahedrite and chalcopyrite, both often altered near the surface into malachite and more rarely into azurite, (3) pyrargyrite and stephanite, (4) metallic silver. The more abundant metallic minerals, such as galena, blende, and siderite, usually occur in association with the gangue or vein matter in interrupted and irregular bands more or less parallel to the walls, as shown in Figs. 21, 22, 23, and 24, whilst the rarer minerals—metallic silver excepted—are most commonly scattered in strings, spots, and small bunches through the former. Metallic silver is generally found in thin flakes along the joints of the quartz. The band-like arrangement of the commoner minerals arises in two different ways, either from one or more of them forming a continuous layer, as on the foot wall of the Whitewater Mine, Fig. 24, or like that on the hanging wall of the Alamo vein, Fig. 25. Or it may be brought about by their being scattered as spots, strings, and bunches through the gangue in such a way as to form an irregular band like that near the hanging wall in Fig. 24.

The metallic minerals are not distributed in all parts of a vein alike, some portions being entirely barren, whilst others are more or less prolific. The ore-bearing parts may consist merely of a few

strings, spots, or blotches of the metallic minerals scattered through the gangue in a disconnected and irregular manner, or the metallic minerals may be concentrated and extensive. In the latter case

FIG. 21.—A horizontal section of vein. (Idaho Mine.)

A, argillite; B, quartz with some argillite; C, galena and a little blende.
Scale 2 in. to a foot.

they are usually known as "ore-chutes" or "pay-chutes." The mines of the Slocan are yet too young for much to be said about their pay-chutes. In not a single instance, perhaps, has the full extent of a pay-chute been ascertained, either horizontally or vertically. Nor do we know anything yet about the pitch of them. In some cases a projection of the stopes may seem to suggest a pitch, but an acquaintance with the structure of the ground shows

that the inclination, of what seems to be the ends of the chute, is due to the existence of cross faults that intersect and shift the vein and are not therefore the natural terminations of the ore-chutes.

FIG. 22.—A vertical section of vein. (Corinth Mine.)

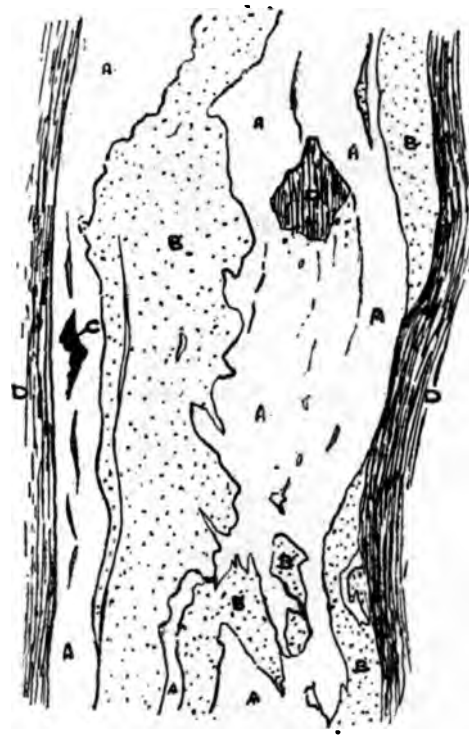
A, argillite; B, galena and limonite; C, calcite and quartz.
Scale 2 ft. to an inch.

This is partly explained by the longitudinal projection in Fig. 13. The area of the pay-chutes opened up in two of the more important mines is shown in Figs. 4 and 11. Since these projections were prepared the stopes have been considerably extended. How large the chutes may prove to be eventually, time alone can tell.

The metallic minerals in the pay-chutes are not often found in any great force throughout the full width of a vein, unless it be narrow. Where wide veins carry ore from wall to wall they usually contain a large proportion of gangue or veinstone. In the majority of cases the ore is concentrated, more or less, along one or other or both of the walls. As seen in section, the portion of a vein containing the metallic minerals is commonly spoken of as the "paystreak." It bears no fixed relation, in extent, to the vein as a whole. The vein may be many feet wide, and the paystreak only a few inches. For example, the Whitewater vein varies in width from 3 to 15 ft., whilst the paystreak on the footwall ranges from 1 to 24 inches only. There is another paystreak near the hanging wall, but, so far as is known,

its extent will not exceed, if it equal, that on the foot wall. In the Ruth vein the paystreak has, in one instance at least, occupied five-sixths of the width of the vein when that was 8 ft. across it, but that

FIG. 23.—Section showing distribution of minerals in vein. (Alamo Mine).



A, quartz (granular); B, blende; C, galena; D, argillite.
Scale half full size.

was quite an exceptional occurrence with a vein so wide. In narrow veins the paystreak may extend from wall to wall, as was the case in parts of the Alamo vein. For example, the vein below the "horse" in Fig. 25 is of this character.

Occasionally the metallic minerals occupy the entire width of a vein to the exclusion of quartz or country rock, except when the latter occurs as "horses." A case of this kind is given in Fig. 26, which is a cross-section of a remarkably well developed part of the Ruth vein. One half of the entire vein—or 4 ft.—was argentiferous galena, without a trace of any non-metallic mineral whatever, and

FIG. 24.—Vertical section of vein. (Whitewater Mine.)

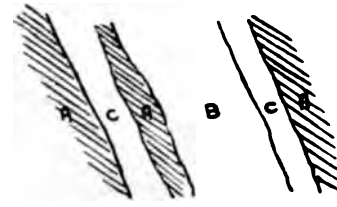
A, argillite; B, argillite and quartz, the former crushed, crumpled, and softened; C, galena and sphalerite.
Scale 20 ft. to an inch.

FIG. 25.—Vertical section showing "horse." (Alamo Mine.)

A, argillite; B, argillite (soft), cerussite, and limonite; C, galena.
Scale 8 ft. to an inch.

a third more was composed of argentiferous galena and siderite combined, in the proportion of one of siderite to two of galena. The remainder (16 in.) of the vein consisted of argillite as a horse

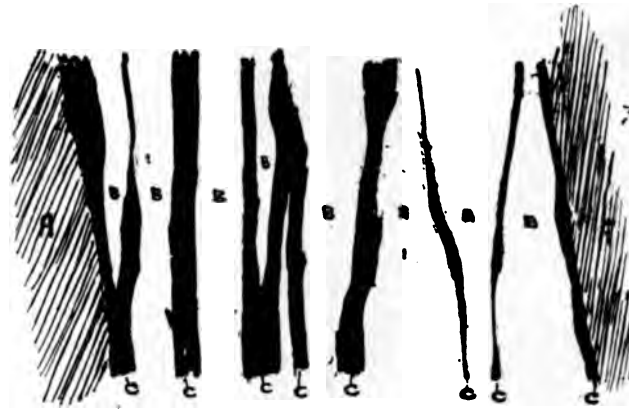
FIG. 26.—Vertical section of vein where unusually large. (Ruth Mine.)



A, argillite; B, galena; C, galena with about one-third siderite and limonite. Scale 8 ft. to an inch.

that is to say, it was entirely free of either metallic minerals or of xenogenous non-metallic minerals. Another section of a vein, composed entirely of metallic minerals and horse, is given in Fig 27,

FIG. 27.—Vertical section of vein in No. 3 tunnel, 900 ft. from "day." (Payne Mine.)



A, argillite; B, argillite, softer than the wall rock A and bed planes mostly obliterated; C, galena with some limonite. Scale 7 ft. to an inch.

which is a section seen in No. 3 tunnel of the Payne Mine at about 960 ft. in from "day." This curious section contains 10 separate and distinct ribs of ore—composed of about two-thirds galena and one-third limonite—having an aggregate width of 50 in., whilst the full width of the vein was 17 ft. Horses often assume the form of

an irregular lens—both in vertical and horizontal section—like that shown in Fig. 25.

The average width of the paystreak in one or two important mines may here be given. In the Payne it may be considered as about 8·3 in. over the whole area of the stopes. In the Alamo as 14·3 in. The paystreak on the foot wall of the Whitewater averages about 7 in. These figures do not, of course, afford a measure of the relative value of these veins over a given area stoped, as that is also influenced by the percentage of metallic minerals they contain. To such an extent is this so that a narrow vein might be much more valuable than one far wider.

When gouge occurs between the ore and its walls they are easily separated in working, but in places, as where there is a fault only on one side of the ore, the latter is separated with difficulty from the unfaulted wall. The ore is then said to be "frozen" to the wall.

Near the surface the metallic minerals have often undergone considerable alteration, galena being changed into cerussite, siderite into limonite, and copper sulphides into carbonates. Fig. 28 gives a

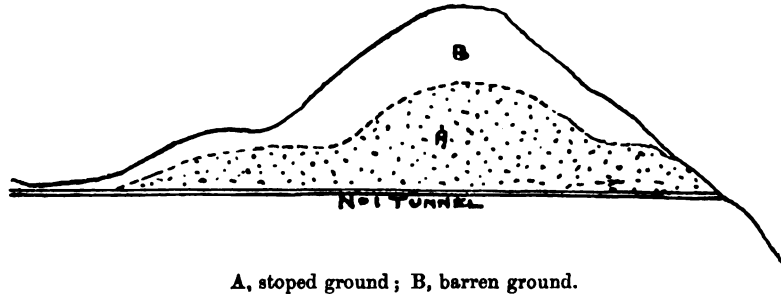
FIG. 28.—Vertical section of vein at surface. (Monitor Mine.)

A, argillite; B, carbonate of lead; C, galena.
Scale 4 ft. to an inch.

section illustrating some of the effects of these changes. The galena has been partly altered into cerussite, kernels of galena still remaining, within the cerussite, as evidence of the change. The lead carbonate is very soft, and absorbs a large amount of moisture. It

will be noticed that the vein becomes narrower towards the surface. Is this an accidental occurrence, or the effect of a cause operating widely? We know that veins have often a tendency to close together when the walls are unsupported. Whether the soft altered ore which now fills the upper part of this vein can, under all circumstances, give the necessary support is not evident, for the ore may eventually, by slow degrees, be either forced out at the surface or removed by meteoric or other denuding agents. The latter is, perhaps, the more probable. In either case the effect would be to produce a narrowing of the vein upwards. It is a matter worthy of consideration whether we have not here a suggestion that will help us to explain a very curious fact, sometimes observed in the Slocan lead veins and in other veins in B. C. and elsewhere. It is not an uncommon observation that a vein which on the surface shows only iron stains and quartz may, after a few feet of driving, begin to produce galena and other metallic minerals, small at first but gradually increasing. The Alamo vein illustrates this point. That vein crosses a "hog's-back," and in stoping above the topmost tunnel the ore was found to nip out entirely some distance short of the surface, and in a line which, although not parallel to the surface, had also the form of a hog's-back, as shown in Fig. 29. In view of

FIG. 29.—Longitudinal section. (Alamo Mine.)



A, stoped ground; B, barren ground.
Scale 200 ft. to an inch.

the facts just recorded, it does not seem improbable that the now barren part of the Alamo vein, adjoining the surface, may have been originally ore-bearing, that the galena was altered and softened as it now is at the Monitor, that the softened ore was removed in the ordinary processes of denudation, and that the walls gradually closed together as the intervening support was removed. This explanation may be applied to other similar occurrences in veins carrying galena, and also to veins carrying other minerals—such as copper ores. It is, therefore, of some interest to the prospector;

when rightly understood it may lead to the discovery of ore bodies in places where the surface indications are not very encouraging.

Although, as already stated, there is usually in the veins a more or less banded arrangement of the metallic minerals, yet the duplicate layers and "comb-structure," so freely illustrated in the text books, are very seldom met with. The prominence commonly given in such works to sections illustrating these somewhat rare phenomena, tends to promote an entire misapprehension as to the inner nature of mineral veins, not only in British Columbia but elsewhere. Fig. 30 is a reproduction of the only instance of duplication and

FIG. 30.—A vertical section of contracted part of vein. (Alamo Mine.)

A, quartz (combed); A*a*, quartz (granular); B, blende; C, galena; D, argillite; E, bitter spar.
Scale half full size.

comb-structure that the writer has met with in the Slocan, and they are exhibited in only one part of the section—the left side. Blende

has been deposited first, irregularly but on each wall, then galena, and lastly quartz, in more or less freely developed crystals.

In some places the metallic minerals follow the lines of jointing in the country rock, branching off from the vein, along both the bed joints and divisional planes, in strings up to an inch or more in thickness. In the Washington Mine it is common to find large strings of galena, without a particle of quartz or other non-metallic mineral in them, running out from the vein into the argillite of the hanging wall. In Figs. 17, 18, and 19 somewhat similar strings of galena are seen to traverse the felspar porphyry. Fig. 31 exhibits

FIG. 31.—Section showing occurrence of blende in felspar porphyry. (Alamo Mine.)

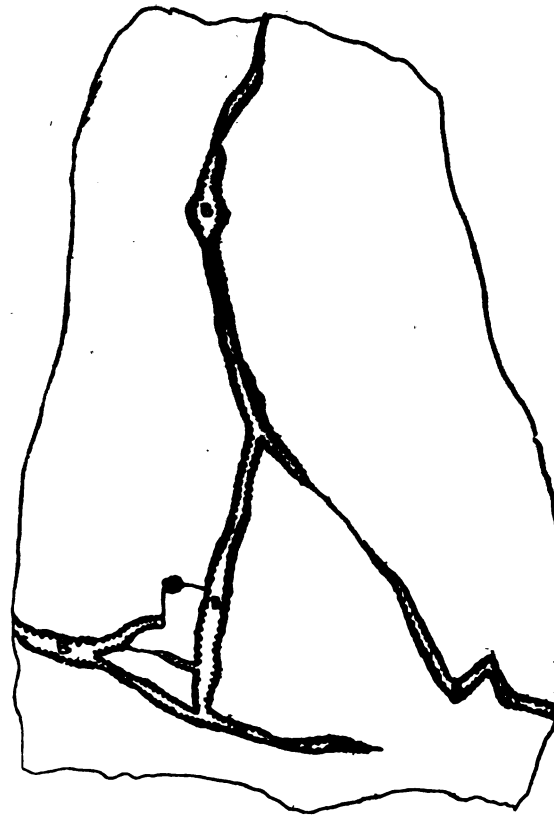
A, felspar porphyry; B, blende; C, ferriferous dolomite. Scale one-fourth full size.

a curious occurrence of blende in that rock. At first sight it might be mistaken for a breccia, but closer examination shows that the blende has most probably been deposited in cavities in the porphyry. Around each nest of blende is a narrow band of dolomite, the inner edge of which is irregularly serrated, as would have been the case if the dolomite had crystallised on the walls of cavities, as in Fig. 32. If we imagine the cavity there shown to be filled with blende, we have a set of facts practically paralleling those presented in Fig. 31.

A fact of exactly the opposite kind to those just mentioned may now be noticed. Most horses are pieces of country rock entirely surrounded by vein-matter, in which the metallic minerals may or

may not occupy an important place, but in Fig. 33 a piece of country rock is seen to be entirely surrounded on the plane of the section by cubic galena. This section also presents another instance

FIG. 32.—Section of partially filled cavity in argillite. (Idaho Mine.)



A, argillite; B, cavity lined with bitter spar.
Scale three-fourths full size.

of duplication. Whether the included argillite was originally attached to the wall-rock cannot be said, as the piece of ore in which it occurred was severed from the vein before the inclusion was noticed. Parallel to the plane of the section the argillite as well as the ore had been broken.

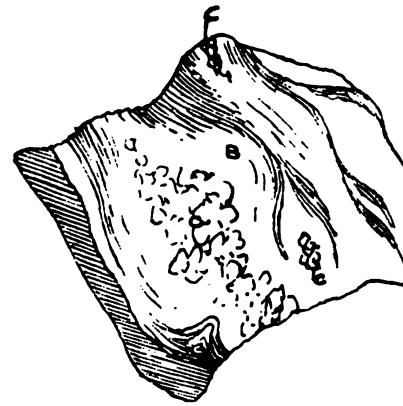
When we come to look closer into the ore, we find some curious facts relating to the order in which the various minerals have been introduced into the vein. Such facts have an important bearing on

the genesis of the deposits, but it is not intended to enter at any great length upon that inquiry here. Figs. 34 and 35 show the manner in which granular and laminated galena are sometimes

FIG. 33. (Washington Mine.)

A, argillite; B, blende with quartz; C, cubic galena with nests of tetrahedrite *d*.
Scale half full size.

FIG. 34. (Alamo Mine.)

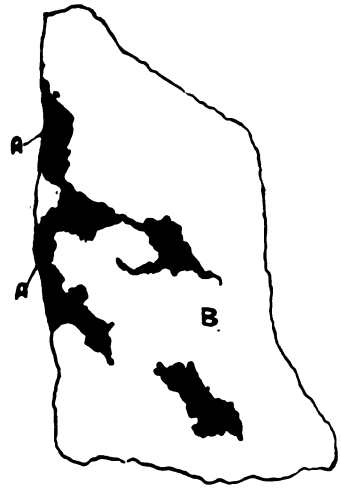


A, argillite; B, galena, laminated where shown; C, quartz.
Scale full size.

FIG. 35. (Enterprise Mine.)

A, blende; B, galena, coarsely granular, Bb laminated.
Scale full size.

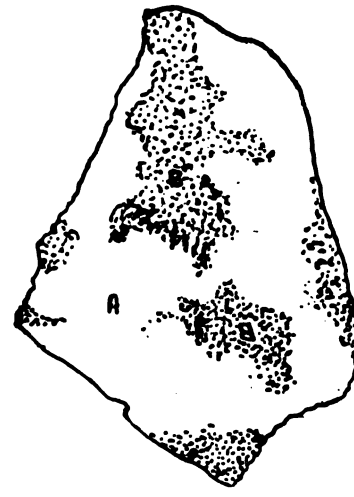
FIG. 36. (Wellington Mine.)



A, galena; B, blende.
Scale full size.

mixed. Fig. 36 shows how galena sometimes traverses blende in a reticulated form. Fig. 37 will explain the way in which tetrahedrite usually occurs in the galena, except that, in this case, the proportion of grey copper is abnormally high.

FIG. 37. (Ruth Mine.)

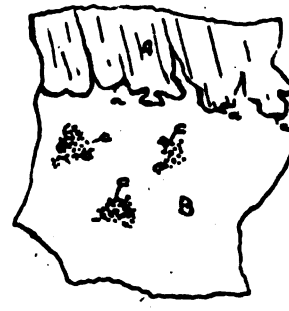


A, galena, coarsely granular; B, tetrahedrite.
Scale full size.

Siderite is of frequent occurrence in the Slocan veins. Originally it was much more abundant than it is now, a large part of it having been converted into limonite, especially near the surface. Much of this iron oxide is shipped to the smelter under the name of carbonates. Some lead is shipped with it, but it is in the form of galena rather than of cerussite, although in some places this latter ore doubtless occurs in the vein, along with the limonite, and will then, of course, be shipped with it. Figs. 38, 39, and 40 illustrate the association of siderite with a variety of other metallic minerals. From the manner in which the galena in Fig. 38 runs into the siderite, it would appear as if the latter mineral had been deposited before the galena. Again, the occurrence of idiomorphic crystals of siderite in Fig. 39 leaves little, if any, doubt in the mind that, in that case at any rate, the iron preceded the zinc. In Fig. 40 pyrite has obviously followed siderite, and yet idiomorphic crystals of the former are bounded by siderite. This section also shows how, in places, tetrahedrite occurs in association with blende. Pseudomorphs of limonite, after siderite, are occasionally met with.

Chalcopyrite appears to have been one of the latest metallic minerals to enter the veins. A curious piece of ore which will illustrate this is shown in section by Fig. 41. This ore occurred in the

FIG. 38. (Whitewater Mine.)



A, siderite, altered to limonite at edge *a*; B, galena (granular); C, tetrahedrite. Scale full size.

FIG. 39. (Wellington Mine.)

A, siderite; B, blende. Scale full size.

clayey gouge, on the foot wall of the Whitewater vein. It was severely striated lengthwise, having evidently been subjected to considerable abrasion, a fact which shows that the Whitewater fault at least (if not others) has had a movement since the deposition of the ore. Fine threads of chalcopyrite, it will be seen, extend from the main body of that ore into both the galena, on one side, and the

blende on the other. Chalcopyrite may, in some veins, occasionally be seen in the form of strings and threads in tetrahedrite.

FIG. 40. (Whitewater Mine.)

A, siderite; B, blende; C, pyrite; D, tetrahedrite.
Scale full size.

Quality of the Ore.—Most silver-lead ores are so associated with gangue, or with the ores of zinc, copper, and iron, that it is not economically possible to ship them in a state of purity, but in a district like the Slocan, which is far removed from smelters, and where, consequently, high freight rates have to be paid, it is imperative to aim at a high tenor. In the early days, *i.e.*, in 1892, when the ore had to be "packed" out to the lakes, only the best ore in a mine could stand the cost of shipping, so that then the ore had to be sorted most carefully. This will be illustrated by a few figures relating to the output of the Dardanelles Mine, one of the first shippers. 314 tons of ore shipped from that mine in two years ending December, 1895, gave 23·7 per cent. lead and 260·9 oz. of silver per ton, but the first 10 tons, packed out during the fall of 1892, yielded 55·6 per cent. of lead and 470·2 oz. of silver per ton.

In most of the Slocan mines some very high grade ore can be found, if only small samples be taken, but that, it need scarcely be said here, is not the way to judge of the product of a mine. The yield of large quantities shipped in the ordinary course of business is the only reliable guide. The following assay results are averages derived from smelter returns relating to the quantities standing opposite the names of the different mines given in the table:—

Name of mine.	Shipments. Tons.	Silver. Oz. per ton.	Lead. Per cent.	Zinc. Per cent.	Silver. Oz.	Lead. Lb.
Payne	11,675	112	56	n. d.*	1 to	10
Whitewater	5,610	113	33	18·8	1 „	5·8
Ruth (galena)	1,010	105	65	n. d.	1 „	12·5
„ (carbonates)	957	48	25	n. d.	1 „	10·4
Idaho	1,531	152·6	46·5	14·5	1 „	6
Alamo	4,782	129	44	16·9	1 „	6·8
Slocan Star	10,012	80	67	n. d.	1 „	16·7
Enterprise	1,054	177	22	21	1 „	2·4
Reco	1,690	239	39	n. d.	1 „	3·2
Washington	1,600	95	56	12·7	1 „	11·7
Dardanelles	255	265	26	12·5	1 „	1·9
Monitor†	427	179	38	15	1 „	4·24
Antoine	230	246	46	n. d.	1 „	3·7
Cumberland	263	78·9	58·4	12·8	1 „	14·8

This table shows better than many words the highly argentiferous character of the Slocan ores. It includes probably the highest and the lowest grade yet found. When we compare these results with those obtained from the ores of other important silver lead areas, the greater richness of the Slocan ores is at once evident.

In New South Wales, at the Broken Hill Proprietary, during six months ending the 31st May, 1897, 145,473 tons of ore were smelted, with the following average yield:—

Silver ..... 22·97 oz. per ton.
Lead..... 8·42 per cent.
Or 1 oz. of silver to 7·3 lb. of lead.

From Block 10, in the same locality, the following quantities and yields were produced:—

Tons.	Silver. Oz. per ton.	Lead. Per cent.	Zinc. Per cent.	Silver. Oz.	Lead. Lb.
1,023	30·0	30	27·5	1 to	20
2,197	31·8	28·4	29·2	1 „	17·8
28,058	19·87	19·28	28·5	1 „	19·4

In the Cœur d'Alene Mines, Idaho, U.S.A., the average, over six years, was 1 oz. of silver to 34·3 lb. of lead.

In Colorado, the ratio of silver to lead in the carbonate ores was as under:—

Tons.		Silver. Oz.		Lead. Lb.
10,561	from Fryer Hill gave	1	to	6·5
6,315	„ Carbonate Hill gave.....	1	„	8
4,794	„ „ „ „	1	„	32
152,457	„ Iron „ „	1	„	26

* Not determined. † Also contained gold, 0·35 oz. per ton.

From the sulphide ores of the same area the following ratios have been obtained:—

	Silver. Oz.		Lead. Lb.
Moyer shaft	1	to	62·8
Colfeller's Mine	1	„	20·3
Minnie Mine	1	„	35·1

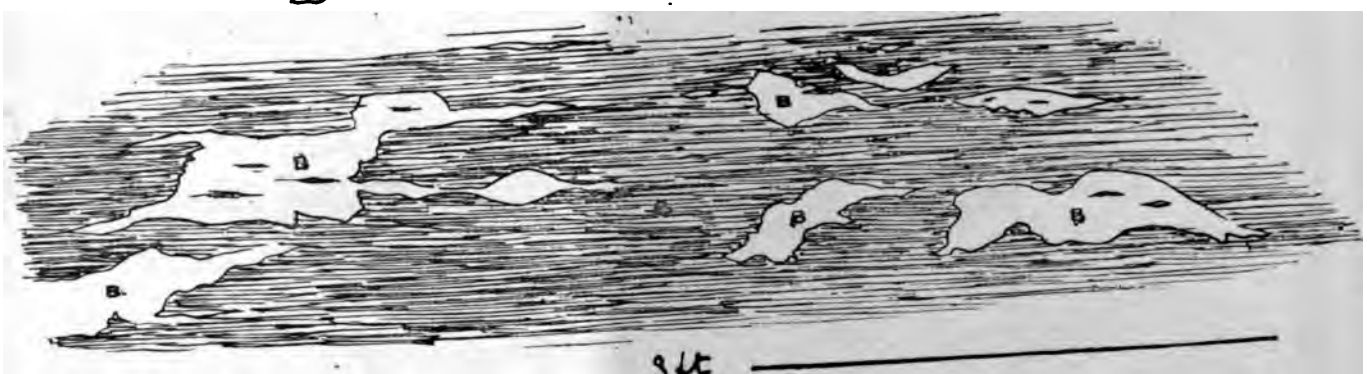
FIG. 41. (Whitewater Mine.)

A, galena; B, chalcopyrite; C, blende.
Scale full size.

The average grade of the ore shipped from the Sierra Mojada, Mexico, was about 30 per cent. lead and 35 oz. of silver per ton, or 1 oz. of silver to 17 lb. of lead.

Returning to the Slocan ores, a further insight into their character may be obtained from a study of the variations among their metallic contents, as shown in the following tables, which give the

Section shewing Quartz lenses occurring along bed-planes of Argillite.



A. Argillite B Quartz

fig 42.

highest and lowest yields of silver, lead, and zinc in carload lots :—

	Silver.		Lead with—	
	Highest.	Lowest.	Highest silver.	Lowest silver.
	Oz. per ton.		Per cent.	Per cent.
Washington	141	75	67	49·1
Monitor	367·6	98·7	32	37
Antoine	386	151	54	45
Dardanelles	470·2	145·8	55·6	15·5
Payne (carbonate) ...	166	61·5	76	32
„ (galena).....	207	109	75·5	30
Ruth „	124	97·5	67·5	62·5
„ (carbonates) ..	73	43·3	31·8	18
Whitewater	370	56	56·4	27·9
Enterprise.....	222·6	74	24	20

	Lead.		Silver with—	
	Highest.	Lowest.	Highest lead.	Lowest lead.
	Per cent.	Per cent.	Oz. per ton.	
Washington	71·2	36·3	123·6	60·1
Monitor.....	54·5	19	304	128·4
Antoine.....	55	13	281	175
Dardanelles	55·6	15·5	470·2	145·8
Ruth (galena)	73·3	62·5	105·8	97·5
„ (carbonates) ...	36·2	16·4	66	45
Whitewater	64·4	11·1	191·0	298·5
Enterprise.....	30	12·8	187·9	161·6

	Zinc.	
	Highest.	Lowest.
	Per cent.	Per cent.
Washington.....	17·5	4
Monitor	23	8
Dardanelles.....	17·5	10·6
Payne (galena)	13	6·57
Whitewater.....	24	4·2
Enterprise	26·2	19

A considerable part of the silver found in these ores is in the galena, which occurs in three principal forms, cubic, wavy, and granular (coarse and fine). From a number of assays of ore from different mines made with the object of learning in which of these forms the silver was most abundant, the following results were obtained :—

	Silver.	
	Oz. per ton.	Average of—
Wavy	165·1	11 assays
Cubic.....	117·4	16 „
Granular (fine).....	109·8	7 „

These figures must not be taken for more than they stand. The variation in the quantity of silver in each of these forms was considerable; but the lowest silver in any of the 34 samples was in the fine-grained ore, and the highest in the wavy ore, which is the same relation as that established by the averages.

Part of the silver resides in the blende. The average of the assay results of six random samples of blende gave 17·6 oz. of silver per ton, ranging from 9 to 35. Some of the blende is cadmiferous. A sample from the Enterprise Mine gave 88·5 oz. of silver per ton, the highest found in the non-cadmiferous being 35 oz. of silver per ton of blende. Sometimes tetrahedrite is intimately mixed with blende, as in the lower part of Fig. 40, then the silver yield is high. Three carloads of this ore shipped from the Wellington Mine, gave the following assay results :—

Tons.	Silver. Oz. per ton.	Lead. Per cent.	Zinc. Per cent.
10	281·5	9·5	42·5
17	320·0	14·6	39·1
18	158·2	15·5	37·2

The silver yield of the ore is also affected by the tetrahedrite which is frequently scattered through the galena. It is further increased by the metallic silver and silver sulphides. The tetrahedrite sometimes carries as much as 6,000 oz. of silver per ton of that ore.

When the metallic minerals are much scattered through the gangue their percentage of the total vein contents is low, as will be seen from the following assay results of mill feeds :—

	Silver. Oz. per ton.	Lead. Per cent.	Zinc. Per cent.
Idaho	74·7	5·4	10·45
Slocan Star	16·21	7·6	—

The gangue of some veins carries as much as 18 oz. of silver per ton when there is not a visible trace of any of the base metallic minerals, but analysis generally reveals their presence, in quantities sometimes exceeding 4 per cent.

Age and Origin of the Deposits.—On the question of age very little can be said that would be more than mere hypothesis, as the geology of the country has not been sufficiently studied, whilst the genesis of ore deposits is a subject much too vast to be dealt with in a communication of this character. But the tendency of suggestion in some of the more prominent features of this new-born mineral district may not unprofitably be considered for awhile if only in outline.

There is no evidence in the Slocan as to the age of these deposits, and perhaps the most that can yet be said as bearing even remotely on the question is that in other parts of the province galena has been found in rocks of triassic age.

Mineral veins, *i.e.*, true veins, are usually supposed to be filled fissures, but there is no evidence of a fissure as ordinarily understood, in the Slocan veins. The greater part of the veinstones consists of country rock more or less altered. Here and there we meet with microscopic evidence that some of the vein contents have been deposited in cavities, as, for example, in Figs. 30 and 31; but such phenomena are rare, and when they occur are invariably on a small scale. Crustification, as Posepny very appropriately called the crystallised lining that forms on the walls of cavities, is an uncommon occurrence.

All the Slocan veins are on the line of one or more faults. As faults they differ in no respect from those which cross the veins. The latter we know are not usually accompanied by deposits of metallic minerals, nor yet by fissures in which such minerals might some day be deposited; but if veins were preceded by fissures that were formed simultaneously with the faults we now find on one or both sides of veins, why should there not be fissures (ready to receive veins) alongside the cross faults? So far as relates to Slocan veins the idea of filled fissures is not permissible. Fifteen years ago the writer showed that it was equally impossible in another area.* During the interval he has examined hundreds of veins in various parts of the world, but has not seen a case where it was either necessary or helpful to assume a pre-existing cavity of the nature of a gaping fissure.

Let us suppose for a moment that all the metallic minerals, all the quartz, and other xenogenous non-metallic minerals occurring in any one of the Slocan veins were removed. Should we have a fissure left? No. We should have a complicated network of cavities traversing the altered country rock in all directions (within the vein space), and varying in width in the different chambers, and sometimes even in the same chamber, from narrow joints—the sides of which are almost touching—to spaces 5 or 6 ft. wide, varying also in length and depth in the several chambers from a few inches to 40 or 50 ft. Let us in this connection look at Figs. 21, 22, 25, 27, and 30, and ask ourselves if the cavities that would be produced by a removal of the metallic and xenogenous non-metallic minerals are such as could be produced by dislocation—whether they are not much rather of the nature of what Posepny calls spaces of dissolution.

* "The Mineral Veins of the Lake District," *Trans. Manchester Geo. Soc.*, 1884.

Microscopic as well as macroscopic study of the order in which the various minerals found in these veins were introduced makes it evident that quartz preceded the metallic minerals, although doubtless in cases of crustification like that shown in Fig. 30 it also followed them. The peculiar forms of the quartz in the vein stones make it also clear that it does not occupy spaces of dislocation. In illustration of this Fig. 42 is introduced. The spaces occupied by the quartz in that figure are obviously more of the nature of spaces of dissolution than of spaces of decission, but they are exactly like many of the quartz forms seen in Slocan veins.

It has been seen that the country rock forming a large part of the vein stone of every vein is, as a rule, very much altered both mechanically and chemically, that the chemical alteration has been effected by the removal of the alkalies and bases. In some places the metamorphism is much more advanced than in others. Let us assume it to be continued until practically all the alkalies and bases are removed; we should then have nothing left but silica, and seeing that such changes usually proceed outward from lines of jointing, and at different rates in different parts, dependent largely upon the varying physical condition of the rock, we should expect the quartz to assume forms resembling spaces of dissolution rather than spaces of decission.

An entire removal of all the minerals except quartz must result in the production of cavities. These may afterwards be filled by other quartz or by metallic or other non-metallic minerals. If vein quartz originated in this way we understand at once why it is absent, or nearly so, from deposits of which the enclosing rocks are calcareous.

As to the manner in which the metallic contents of veins were deposited we shall not stop to inquire here; the evidence seems, however, to warrant the statement that they were introduced at different times and perhaps in this order—siderite, blende, galena, pyrite, tetrahedrite, and chalcopyrite; that in some cases they were deposited in cavities like the galena and blende in Fig. 30, in others they were introduced metasomatically like the chalcopyrite in Fig. 41. In a few cases there appears to have been an alternation in the deposition of the metallic minerals, but they are usually on a small scale and such as are seen in most mineral veins, resulting, it may be, from the solution and reprecipitation of minerals that have already existed in the vein. These processes we know are going on in veins to-day on a small scale.

Working the Deposits.—The facilities for mining in the Slocan are great—a good climate, abundance of timber, the contour of the

ground such that the deposit can be reached and worked by tunnels without the necessity for shafts and their accompanying costly machinery, a moderately hard country rock, and abundance of surface water for dressing purposes, in addition to numerous fine veins of high grade mineral, forming altogether a combination of circumstances that could not easily be excelled in any part of the world. The greatest drawback hitherto has been absence of the means of transport, but by the construction of railways and other methods of communication this difficulty is being rapidly overcome. The erection of local smelters to reduce still further the cost of transportation will complete the union of conditions needed for working the deposits in the most economical way.

There has been a proportionately large amount of bad mining in the district, mainly because men have undertaken the work who had not the necessary preliminary training. Out West men seem to think they can do anything, mining included, without previous preparation. Railway men (of whom there are a number in charge of mines) seem to persuade themselves that driving a tunnel between two points is somewhat akin to mining, if indeed it is not mining. As a result we see such men pushing through the ground regardless of geological structure, and making tunnels, but not mines. For similar reasons we sometimes see costly dressing mills being erected where there is not sufficient ore blocked out to pay for a coffee-mill. Unfortunately too these mills are sometimes erected here, as elsewhere, with an entire disregard of the appearance of the mine and for the sole purpose of booming shares.

The usual method of working the Slocan ores is shown in Figs. 4, 11, and 13. It is practically the same as that employed everywhere in like conditions, so that there is no necessity to describe it further here.

Dressing and Marketing the Ore.—When the ore reaches the surface it is concentrated either by hand or by means of machinery. The hand-sorted ore is usually referred to in the smelter returns as "crude ore," that which has been machine-sorted as "concentrates."

The machinery employed in dressing is of the usual kind. There is often considerable loss of silver with the tailings, which is unavoidable and can only be saved by treatment in other ways. So long as the silver-bearing minerals thrown off from a mill have a value less than the cost of freight and treatment, &c., they are properly rejected. The silver so thrown away may afterwards be recovered by other methods and in a concentrated form that can be shipped at a profit. Close dressing will become more of a necessity with the introduction of local smelting.

The first dressing* mill erected in the Slocan was the Alamo, near Three Forks, which was built in 1894. Since that time several others have gone up, as shown in the following table:—

Mill.	Capacity per 24 hours. Tons.	Erected.
Alamo	100	1894
Slocan Star	180	1895–6
Washington	50	1896
Noble Five	130	1897
Whitewater	100	1898

When dressed, the ores are mainly sent to one or other of the United States smelters. A little ore has been sent to the Hall Mines smelter, at Nelson. Recently the Canadian Pacific Railway Company erected a smelter, at Trail, for the purpose of dealing with the Slocan ores, but it has not got fairly to work yet. The heavy freight to U.S. smelters—ranging from $11 to $19 per ton—has hitherto been a serious item of cost. Local smelting will greatly reduce this charge, but it will not obviate the payment of duty unless a market can be found for lead outside the United States. If such a market were found the base bullion could be sent to the States (in bond) for refining until such time as a refinery be built in B.C. The present duty on lead in Slocan ores, entering the United States, is 1·5 cents per lb., and on the lead in base bullion 2·125 cents per lb. The average lead contents of the ore shipped from the Slocan has been about 50 per cent. The duty on ore of this grade, at present rate, is $15 per ton. On a ton of base bullion it is about $42 per ton. It takes over two tons of average Slocan ore to make a ton of lead. The duty on that ore is about $31·80, against $42 if the same weight of lead be shipped as base bullion. Take the freight on ore at $15 per ton (of ore) and the cost per ton of lead must be about $31. Let us see now how matters appear from the standpoint of the Canadian Pacific Railway Company, even with the U.S.A. as a market.

Smelting Slocan Ores in the United States.

	Per ton of base bullion.
Freight at $15 per ton of ore	31·00
Duty at 1·5 cents per lb. of lead in ore	31·80
	$62·80

* In the report of the Minister of Mines for 1896, it is stated that the Washington was the first concentrator erected, but that is an error.

Smelting Slocan Ores at Trail.

Freight on ore and bullion ..... say	18·00
Duty at 2½ cents per lb. of lead in base bullion..	42·00
	$60·00

The difference is in favour of smelting at Trail, even if the base bullion be sold in the United States. If it were sold elsewhere and simply refined in the States (in bond) the benefit to B.C. would be very great indeed. But this is only so long as the operation is in the hands of the Canadian Pacific Railway Company. The $18 per ton charged for freight in the second of the above statements is a figure which can of course be regulated by the railway company to suit themselves. If they were hauling base bullion, for others, doubtless they would demand a much higher rate than for hauling ore, but for themselves they need not do so. They will, in smelting at Trail, lose the long haul of the ore, but as much of the total ore-product now goes over other railway systems they would probably not be any losers ultimately, for they would get the carriage of the whole of the bullion, which would probably much exceed in quantity the ore now hauled by them. Apart from that they would create a great industry which must benefit their system greatly in other ways. No one else (unless given special railway rates) can deal so satisfactorily with this problem, at present, as the Canadian Pacific Railway Company, and it is to be hoped their endeavours to develop the resources of the country in this way will meet with complete success.

Statistics of Output.

The figures given below are taken from the report of the Minister of Mines. Those relating to the first three years are for the whole Province, and are, therefore, in excess of the Slocan output by about 60, 45, and 20 per cent. respectively. The figures for the last three years are for the Slocan Mining Divison only, excluding that part of the Ainsworth Mining Division which has been included in other parts of this communication.

	Ore. Tons.	Gold. Tons.	Silver. Oz.	Lead. Lb.
1892	..	..	77,160	1,768,420
1893	..	..	227,000	2,135,023
1894	..	..	746,379	5,662,523
1865	9,649	6	1,137,040	9,751,464
1896	16,560	152	1,954,258	18,175,074
1897	33,567	193	3,641,287	30,707,705

The rapid increase of output which has taken place and which will doubtless be continued with the growing facilities for operating must, ere long, place the Slocan in a leading position among silver-lead producers. When to that be added the output from other parts of the Kootenay which will, almost certainly, increase at a similar rate, there is presented a very bright prospect for the silver-lead industry of British Columbia.

NINTH ORDINARY MEETING, 21st June, 1899.

Mr. J. H. COLLINS (*Past President*) in the Chair.

THE Minutes of the previous meeting were read and confirmed.

"THE SILVER-LEAD DEPOSITS OF THE SLOCAN, BRITISH COLUMBIA." By J. D. KENDALL, M.Inst.M.M.

DISCUSSION (*continued*).

The Chairman considered that Mr. Kendall's contribution was a very valuable one, and it was a matter for regret that the author was not able to be present. It would have been interesting to have had some specimens of the rocks and minerals before them, but the paper clearly indicated that in the Slocan district there was the well known association of slag slates intruded upon by both basic and acidic eruptive rocks. It also appeared clear that on the whole the veins were what was commonly known as "fissure" veins, although they were not of such a decided type as they were familiar with in the West of England, which were traceable for three or four miles in length; in the Slocan they were only noticeable for a quarter or a third of a mile. The author pointed out that the veins all occurred on lines of fault, and had all been subsequently faulted, and for the most part were newer than most of the eruptive dykes, because they were seen, as in Cornwall, to cut through the eruptive dykes in almost all cases. It appeared to him that there were certain inconsistencies in Mr. Kendall's paper, although no doubt if that gentleman were present he would be able to explain them. Mr. Kendall said that "mineral veins, *i.e.*, true veins, were usually supposed to be filled fissures, but there was no evidence of a fissure, as ordinarily understood, in the Slocan veins"; and also that "so far as related to Slocan veins the idea of filled fissures was not permissible." Those remarks did not agree with other statements in the paper, and it seemed difficult to suppose that, in a country where limestones were not noticeable, the greater part of the mineral deposits was due to what the author called the "theory of

spaces of dissolution." They were familiar with deposits that were evidently occupying spaces of dissolution in limestone countries, but it was hard to understand how there could be such spaces in the Slocan. It appeared to him more likely that the mineral deposits consisted of infiltrations into the fractured zone originally due to fissuring and faulting. He thought that the author had read into the term "fissure" something that was not usually meant, when he stated that there was no evidence of anything of the kind in the Slocan veins, because the fissure origin of lodes by no means excluded or was antagonistic to the idea of very extensive chemical action and very extensive mechanical action. Given universal faulting and combs of minerals in some of the veins, gossans, and breccia, it was very hard to suppose they were not real fissure deposits. This really appeared to be the author's own opinion, as would be observed from the following extract from the paper, in which he assumed the fissure deposit idea, and theorised upon it, to account for noticeable phenomena:—

"It will be noticed that the vein becomes narrower towards the surface. Is this an accidental occurrence, or the effect of a cause operating widely? We know that veins have often a tendency to close together when the walls are unsupported. Whether the soft altered ore which now fills the upper part of this vein can, under all circumstances, give the necessary support is not evident, for the ore may eventually, by slow degrees, be either forced out at the surface or removed by meteoric or other denuding agents. The latter is, perhaps, the more probable. In either case the effect would be to produce a narrowing of the vein upwards. It is a matter worthy of consideration whether we have not here a suggestion that will help us to explain a very curious fact, sometimes observed in the Slocan lead veins and in other veins in B. C. and elsewhere. It is not an uncommon observation that a vein which, on the surface shows only iron stains and quartz may, after a few feet of driving, begin to produce galena and other metallic minerals, small at first but gradually increasing. The Alamo vein will illustrate this point, that vein crosses a 'hog's-back'; and in stoping above the topmost tunnel the ore was found to nip out entirely some distance short of the surface, and in a line which, although not parallel to the surface, had also the form of a hog's-back, as shown in Fig. 29. In view of the facts just recorded, it does not seem improbable that the now barren part of the Alamo vein, adjoining the surface, may have been originally ore-bearing, that the galena was altered and softened as it now is at the Monitor, that the softened ore was removed in the ordinary processes of

denudation, and that the walls gradually closed together as the intervening support was removed."

He thought most of them would be disposed to accept that explanation, and they could only regret that the author was not present to explain the apparent contradiction to the statements he had referred to in the latter part of the paper.

Mr. E. P. Rathbone said that last summer he visited the "Slocan Star," the "Whitewater," the "Ruby," the "Payne," and one or two other mines in the district, and one of the most noticeable things in the schists was that the leady deposits seemed to be very weak in places and were without any true walls; wherever there was any movement it did not seem to require much to shift it from side to side—in fact, it seemed to be "weak-kneed" as compared to any granite, or other hard formation. In the "Whitewater" the schists were of a soft muddy character. In other respects the economical conditions throughout the country appeared to be very favourable indeed. He was much struck with the strong declivity of the mountains in the district, which made it possible to work by means of adit levels. The Americans, who were first on the field, had achieved a considerable amount of success by reason of their economical methods of working. The properties were owned by small corporations, and by shipping the ores direct from the mines to the smelters great successes had been made, notably in the case of the "Payne." The principal factor in keeping the district back had been the very heavy freight charges and the terribly heavy duty on the lead sent into the United States. If those conditions could be ameliorated the exceedingly rich character of the ore would allow of very successful work by small corporations. He did not think there were many properties in the Slocan that could be worked on a very large scale; if formed into companies they would require a nominal capital of about £150,000 with £40,000 provided for working capital, and with careful management they would yield a good return on the money invested. One drawback hitherto had been that the management had had to make dividends as quickly as possible, with the consequence that the properties were "gutted," and no development work was done. The country was wonderfully timbered, and there were usually good mountain streams, where sufficient water could be obtained for concentrating. Most of the propositions required good concentrating plant, and many excellent ones had been erected. He considered the district was well worthy of the attention of capitalists for the formation of small, not over-capitalized, companies.

CONTRIBUTED REMARKS.

Mr. W. Thomae: The paper is a very interesting description of the characteristics of the silver-lead lodes of the Slocan district. I should like to have seen a little more in it about the nature of the igneous dykes, touched upon on page 285, and the way in which they affect the mineral contents of the veins.

When I visited the mines in 1895, it struck me that there was considerable variety in their composition. Some were felspar porphyries, others quartz porphyries or quartz felsites, while that at the Alamo Mine was much more basic in character, and seemed to me like a much altered diabase. It is interesting to note that while the acid dykes were locally supposed to impoverish the lodes they came in contact with, that at the Alamo was said to increase the silver contents of the lode passing through it.

It would have been interesting to know whether subsequent observations have thrown any further light on this subject.

The statement on page 293 is significant. The author says :— "In not a single instance, perhaps, has the full extent of a pay-chute been ascertained, either horizontally or vertically."

Lodes that carry such extensive pay-chutes, and that sometimes show a width of 4 to 6 ft. and more of solid galena in the face of a drive, can scarcely, I think, be called small, as silver-lead lodes go, and, combined with their wonderful richness in silver, amply justify the author's concluding remarks.

"The Latest Type of Mechanical Calciner."

By W. Blackmore, Assoc.Inst.M.M.

The Godfrey Mechanical Calciner which is the subject of the present paper, was primarily designed to calcine zinc and leady zinc ores. Hitherto mechanical calciners have not been considered suitable for such ores, and the universal practice has been to calcine both lead and zinc ores, and their mixtures in hand calciners.

English and Continental smelters have always used hand calciners for these ores, and at the present day they are calcining them in pretty much the same way as their grandfathers did two generations ago. Some slight improvements have been made in hand calcination, but not to materially lessen the cost. In America and Australia attempts have been made to introduce mechanical calciners for the lead and mixed sulphide ores, but hitherto these have been only very partially successful, and mechanical calciners have been discarded for such ores.

The reason for this state of things is, that the usual type of mechanical calciner is designed for calcining ores which yield their sulphur with great readiness. For such a purpose they are in many cases admirably adapted, but when we come to the more difficultly calcinable ores, the mechanical calciners have failed. This is undoubtedly to a large extent due to the fact that the extra heat needed for the calcining of certain ores cannot be readily put into the ordinary mechanical calciner. Thus it comes that only those ores which need very little external heat, are able to be calcined in them. It is also safe to say that even if it were possible to introduce a sufficient calcining heat for zinciferous lead ores into these mechanical calciners, the calciners themselves would not be able to stand that heat for any length of time, and would undoubtedly break down. The writer has had some considerable experience in the calcining of the mixed sulphide leady zinc ores. He has had the unfortunate experience of trying to calcine these ores in a four-hearth furnace, each hearth being 75 ft. long, and which had two sets of travelling ploughs. Four more fireplaces than the inventor considered necessary, were not sufficient to bring the furnace to a calcining heat, the result being a complete failure. Near the fireplaces only could any proper heat be obtained, and at these places the expansion of the ironwork was such as to cause incessant trouble.

In the design of the Godfrey calciner a new departure altogether has been made from all existing types of mechanical calciners. It has been so constructed that sufficient heat can readily be obtained for the calcination of any class of ore whatever. At the same time the design is such, and the furnace is built in such a manner, that it suffers no appreciable detriment by the introduction of a large amount of heat.

On examining the accompanying drawings, it will be seen that the calciner consists essentially of an annular rotating hearth. The roof covers all the hearth with the exception of one sector, each portion of the rotating hearth coming in turn into this open sector. In this open space is placed a plough, the blades of which can be moved by a simple lever arrangement in any direction. Looking at the drawing in detail it will be seen that the hearth of the furnace is built on cast-iron cantilevers, of which there are 10, firmly fixed into central rings, and thus forming one rigid frame. On this frame are laid cast-iron plates, and on these again is the actual floor of the furnace, made of 2 in. fireclay slabs. The brickwork roof is thrown between the outer ring of masonry and the central column. The furnace is about 20 ft. in diameter, and as will be seen from the drawing, the frame of the hearth is supported on ball bearings, upon which it revolves, being driven by the ordinary tooth wheel gearing. Only 1 H.P. is necessary to work the furnace. The opening in which the plough is fixed is 3 ft. 6 in. wide on the outside, tapering down to 2 ft. in the centre. The fire hearth on the plan is apparently large for such a furnace. Of course it would be much smaller if the ore to be calcined were of a more kindly nature than the mixed lead and zinc sulphides. Into the furnace the ore is charged through a hopper on the roof or a door in the outer wall, as may be deemed most convenient. The rate of travel is one revolution in from two to three minutes, but it is possible to increase or decrease this speed at pleasure.

In a very short time after the ore is charged it gets uniformly red hot, for each part of the ore charge passes in its revolution through the hottest portion of the furnace. This is one of the great advantages of the furnace, the ore being uniformly heated, one portion is not brought to a much greater heat than is another, as is the case in hand calciners, and in the ordinary mechanical calciners. During each revolution the ore is adequately turned by the plough, and thereby brought into actual contact with pure air undiluted by the furnace and calcination gases. Every two or three minutes each part of the ore is brought into this open space, and exposed to a large amount of oxygen, thereby materially assisting the rapid

calcination of the ore. Thus, the furnace is one in which the heated ore is brought alternately first into the closed portion of the furnace in contact with the furnace gases, and secondly, into the open part away from the products of combustion altogether. It is obvious that more rapid calcination is bound to take place than when the ore is always surrounded by the products of combustion, and sulphur gases which result from the calcination. It is this new principle of bringing the ore into the open air, as it were, to be rabbled that constitutes the novelty, and undoubtedly contributes to the great success of the furnace.

The ploughs being placed in the opening of the furnace naturally are subjected only to a comparatively low degree of heat. This is of itself a very decided advantage, for the repairs to ploughs in ordinary mechanical calciners is a very serious item of expense.

The plough, besides, is so arranged that it can be taken out and have new blades fixed in a very short time. When calcined sufficiently the ore is discharged automatically by moving the blades of the plough against the ore, and in the course of a very few minutes the ore is all withdrawn from the furnace. This operation is effected without cooling down the furnace at all, and the next charge is introduced without delay on the still fully heated hearth.

It may be of interest to state how the cost of calcination by this furnace compares with other methods of calcination.

Firstly, in considering the ores for which the furnace was specially designed, that is to say, zinc ores and zinciferous lead ores, the author has been at some trouble to ascertain the cost of roasting such ores in the Swansea district and on the Continent. He finds that ordinary zinc blendes cost from 12*s*. 6*d*. to 15*s*. per ton according to the nature of the ore and the practice of the different works. Of course in this sweet calcination is understood, that is, calcination down to 2 per cent. sulphur. For zinciferous lead ores the cost in the Swansea district is about 7*s*. per ton, the calcination being carried down to 7 per cent. sulphur. If it be desired to calcine sweeter the cost is about 1*s*. more for each unit of sulphur below 7 per cent. Thus to calcine down to 5 per cent. in hand calciners would mean an expenditure of at least 9*s*. per ton.

By means of this calciner the costs are reduced to a comparatively small figure. A saving is effected of from 70 to 80 per cent. in the costs of hand calcination, for zinc blendes are calcined in it for less than 5*s*. per ton down to 2 per cent. of sulphur. Lead and leady zinc ores are calcined for the small sum of 2*s*. 3*d*. per ton down to 5 per cent. sulphur. In these figures the full costs are allowed

of labour, fuel and power, and repairs, coal being 7*s.* 6*d.* per ton and labour 4*s.* per day.

In addition to the saving in the actual cost of calcination, there is another matter which all lead smelters especially will appreciate, and that is the loss of lead and silver by volatilisation. In the hand calciners there is known to be a loss by volatilisation of from 1 to 2 units of lead, but in this calciner the author has failed to detect any loss whatever of this nature. He attributes the cause of this to the fact that the ore in the furnace is at one uniform heat, and that no portion is allowed to get overheated near the bridge of the furnace, as is the case in the ordinary reverberatory. The rapidity of the calcination is also an element in reducing the loss by volatilisation, as the ore is only in the furnace one-third the usual time required in hand calcination. The saving in this respect thus effected over the hand calciners will certainly pay all the cost of calcination of the ore. In zinc ores this calcination loss is not so apparent, but it is safe to say that there can be very little calcination loss whatever in calcining in this mechanical furnace.

Another matter which is also of some importance is that there is no appreciable soaking of lead or lead oxide into the furnace bottom while the ore is in the furnace. This of course is impossible as there is only the 2-in. thickness of fire slabs forming the hearth, and the ore is calcined perfectly dry.

A 20-ft. furnace will readily calcine 5 tons of ordinary zinc blende in 24 hours, or very nearly the out-put of three ordinary hand calciners. Of leady zinc ores and lead ores it will calcine 10 tons per day down to 5 per cent. sulphur. The furnace will of course calcine much larger quantities of more kindly ores and at a lower cost. Ferruginous and cupriferous pyrites could be calcined probably for 1*s.* to 1*s.* 6*d.* per ton, and each furnace would put through at least 15 to 20 tons per day.

It will be seen from the drawings that the cost of erecting this type of furnace is comparatively small. The weight of the ironwork is about 25 tons per furnace, and 7,000 common bricks and 8,000 firebricks are required for the masonry.

It is important to have good concrete foundations to avoid any chance of sinking of the ball-track and to prevent jamming of the tooth gearing. After the foundations have been laid a furnace can be built up ready for firing in two weeks, or considerably more quickly than a hand calciner that would not calcine half the quantity of ore per day.

Two of these calciners are working at Messrs. Fry, Everett, and

Company's works at Swansea, and a third is in course of erection. The Smelting Corporation are engaged in putting up twelve at the present time.

DISCUSSION.

The Chairman considered that Mr. Blackmore's paper was a very practical one. He had personally heard of the good work the furnace had been doing at the works of Messrs. Fry, Everett & Co. on extremely difficult ores. It remained to be seen whether, as the author seemed to think, it would do better than the existing furnaces on ores that were easier to treat, although he did not see why it should not do so. The only thing that made him think it would not was that it was hardly likely that one type of furnace would prove to be the best for all kinds of ore. It would be interesting if the author could give them some more details of the construction of the furnace, and would let them know how much ore had already been treated in that type of furnace.

Mr. Walter McDermott said that the paper was an interesting one, as it brought forward a type of furnace for which distinct advantages were claimed on actual working results; if such results had not been given it would, at first sight, have appeared questionable whether such an efficiency could be obtained with that type of furnace. One advantage claimed was the moving of the ore into the open air while hot and rabbling it there, otherwise it was something like the old revolving furnace used for tin ore in Cornwall, known as the Brunton. If the open air space was proved in practice to give a higher efficiency with better results, it was one of those small matters which made all the difference between success and failure. He did not agree with the author's statement as to mechanical calciners not having been successfully used on zinc ores in the past, because they have been in successful operation in America for a number of years; the results appeared to be equally satisfactory as far as economy was concerned, and with certainly as great a reduction in the sulphur. In Missouri three furnaces had been in steady use for about three years; they were of the Browne type, in which the ordinary zinc blendes were reduced to an average of 1 per cent. of sulphur.

It was stated that the calciners would not be able to stand the heat for any length of time, even if they did the work; that was not in accordance with the fact that they had been running for several years continuously.

One of the advantages claimed for the furnace was that the ore was uniformly heated; in the mechanical furnace the whole body of the ore was moved gradually forward, and in turn exposed to the same uniform degree of temperature, but was not kept up to that temperature, as in the case of the present furnace working by charges instead of continuous operation. The new principle referred to in the paper of rabbling in the open air would have the advantage of keeping the ploughs cool, but in the usual mechanical furnace the ploughs were taken out, and they had time to cool off, and it was found in actual practice that there was not excessive wear.

Mechanical furnaces had been in use in Wales on lead, copper, and zinc ores for two or three years, and Mr. Christopher James had stated that his experience was that very small repairs were necessary, the blades of the rakes being changed only once in nine months. He could not say that there might not be a greater rate of wear in the case of zinc ores alone, where greater heat was employed, but at all events it was not found to be excessive.

With regard to costs, they compared very favourably with the figures given in the paper, being in the neighbourhood of 2*s.* or 3*s.*, and not 12*s.* or 15*s.* as mentioned by Mr. Blackmore.

It struck him that one of the disadvantages in the construction of the furnace was that the working parts, being underneath the hearth, were exposed to a greater amount of heat than was the case in other mechanical calciners. In addition, the radiation of the heat from the hearth made a great difference in the consumption of fuel —a solid hearth well packed underneath would retain the heat and use much less fuel than a hearth with many openings and chances for the radiation of heat.

With further reference to costs, Mr. C. James mentioned that he had been hand-calcining some ores the cost of which had been 6*s.* 8*d.* per ton, while using a mechanical furnace the same ores had been treated for 3*s.* per ton.

He should like to know whether the furnaces had been running for a sufficient length of time to settle the points he had raised; he should imagine, however, from the erection of 12 additional furnaces that the people using them were quite satisfied with them in practice.

Although the cost of the furnace was small the capacity, as compared with the big mechanical furnaces, was also small, and it would require three of them to do the work of one ordinary mechanical roaster, so that the difference in cost was not so apparent when taken on the tonnage of ore to be treated.

Professor Huntington considered that the idea of exposing the

ore to the air without the presence of the gases was a good one, although on the other hand the fact that it was exposed to cold air would probably not be altogether an advantage, as the ore would be cooling down.

The author made rather a sweeping statement when he said that the mechanical furnaces in use in America were not efficient. There was good reason to believe that several of the mechanical furnaces at present in use were doing good work.

With regard to heating the ore uniformly for a sufficient length of time, he did not see any difficulty with the present furnaces—it was only a question of travelling the ore through a sufficient length of furnace.

Although it was possible that good work might be done with the furnace described by Mr. Blackmore, he did not think that sufficient evidence had been brought forward on which to condemn the large mechanical furnaces already in work. He took it that the furnace was limited as to size and could not be extended as was the case with the American mechanical furnaces.

Mr. Lockhart asked whether there was any attempt to make a joint between the furnace and the space below, as it seemed to him that there would be a good deal of cold air coming in constantly which would cool the furnace very much. He agreed with Mr. McDermott that the ironwork underneath the hearth must be very hot, and wished to know how the balls behaved, and also whether it was possible to lubricate them while running.

Mr. Blackmore, in replying to the points raised, said that, in answer to the chairman's question, there were at present only two of the furnaces working—those had been in operation continuously for over three months, and had given no trouble. They had calcined some hundreds of tons of leady zinc and zinc blende ores.

With regard to Mr. McDermott's remarks as to the American furnace, he had spoken to zinc smelters in the Swansea district and they stated that it was not a furnace they would care to adopt as they did not consider it suitable for pure zinc blendes. The fact that the Missouri furnaces had not been adopted in any English or European works indicated that they were either unsuitable for ordinary zinc ores or that European smelters were afraid of them. He believed that Mr. McDermott was in error when he stated that mechanical calciners had been in use for lead and zinc ores in Wales for several years. As far as the author was informed no zinc ores had been calcined in the furnace to which Mr. McDermott referred, and copper ores and copper mattes were the materials chiefly calcined in it. Mr. McDermott's statement of the reduction in

costs by such a furnace did not therefore hold for zinc and lead ores for which the Godfrey calciner was designed.

They did not find any great loss of heat by radiation from the ironwork of the furnace. Indeed the small quantity of coal consumed in it was one of its great advantages. A 20-ft. furnace required 30 cwt. of coal per day, which is much less than a reverberatory furnace that would calcine one-third the quantity of ore.

The heating of the running gear was very little, and they were able to lubricate the balls properly with very little consumption of oil. The balls run very smoothly with very little noise or friction.

With regard to Professor Huntington's remarks, it was possible that hot air, if supplied in sufficient quantities, would be an advantage, but it would certainly be an additional expense. The loss of heat at the opening was not appreciable, for the process of calcination which took place at the opening most rapidly was itself heat-producing. He did not think that the limitation of the size of the furnace was an objection. It did not matter to a smelter whether any given calciner calcined 50 or 10 tons per day—it was the cost per ton which was the all-important point. As a matter of fact, however, all the calciners which claimed to calcine such large quantities of ore per day, referred only to soft ores, not to lead or zinc ores. Ten tons per day was a considerable quantity for one calciner to work effectively of mixed lead and zinc sulphides.

The author would prefer to have several small calciners working in preference to one large one, provided the cost of calcination were the same in both instances, because in case of the breakdown in a large furnace, the whole operation of smelting might have to be suspended—a contingency that would not arise in the other case.

In conclusion, the author speaking from his experience with the Godfrey calciner, was strongly of opinion that after a longer period of working, he would be able to inform the members of the Institution that there were very few repairs necessary, and that the life of a furnace was such as to bring the capital charge to a very small item per ton of ore passed through.

Mr. McDermott remarked that the fact that the furnace he had mentioned was not adopted in Swansea, was not conclusive evidence against it. They did not necessarily adopt furnaces in Swansea that were in use in other parts of the world.

The fact that there was a considerable saving of fuel over the hand-roaster was no argument as against other mechanical furnaces, because exactly the same thing occurred, as mentioned by Mr. Christopher James, the saving was very large in the mechanical furnace as compared with the reverberatory. His only criticism on

the paper was as regards the statement as to the failure of other furnaces, while pointing out the advantages of the one brought before them. It seemed to him that the comparatively short time in which the furnace had been running, left a good many questions to be settled as to its superiority.

"Notes on an Improved Muffle Furnace for Burning Coke."

By T. Graham Martyn, Assoc. Inst.M.M.

The Coke Muffle Furnace, as employed at the School of Mines, London, the Mining School, Camborne, and probably in many assay offices all over the world, is almost exactly similar to a "Wind Furnace" with a hole left in the front, about 6 in. above the bars, through which the muffle is inserted. The muffle reaches to within an inch or less of the back of the furnace, and is supported there by a projecting piece of brick. If a long one it is sometimes supported also at each side near the front, but such projections interfere with the passage of the fuel. There are several disadvantages in this arrangement, which may be stated as follows:—

1. Coke fed on the top of the muffle. Fine dust accumulates there and clinkers badly.

2. The space at the sides, usually about 2 in., is too restricted and frequently chokes more or less completely. This necessitates the use of the poker at great risk to the muffle.

3. The draught passage at the back gets choked the first day of use and cannot be cleared. This makes it necessary to keep the mouth of the muffle open for the free access of air, unduly cooling the front part.

4. The coke feeding down on the fire bars at each side must be frequently raked to the middle, or else cold air will rise between the middle bars in direct contact with the bottom of the muffle.

5. The connection with flue being at the back of the furnace the draught is concentrated in that direction especially at the top

The greatest heat is thus applied to the back of the muffle which is at the same time furthest removed from, and least liable

to, sudden chill by draughts of cold air. Hence the temperature of the muffle is very uneven from front to back, and the top of the muffle never gets thoroughly heated at all.

6. The muffle is not efficiently supported, being liable to crack across the middle and, weighed down by the coke on top, it may fall away without warning.

7. A sliding damper is frequently employed, which is rather awkward to work. If thin it is liable to warp and jam, and if thick it is very clumsy to move.

The diagrams accompanying these notes give a longitudinal and a transverse section of a furnace in which the above defects are almost completely remedied.

1. The coke is prevented from resting on the top of the muffle by the single-brick walls, GG, of the flue opening H. A little dust may settle on the top, but practically no clinker is formed, so that the transmission of heat is not interfered with.

2. The space, KK, each side of the muffle is increased to 4 in. being wider than that, LL (3 in.), through which the fresh coke is fed, hence no obstruction is caused, even if it becomes slightly clinkered up.

3. The draught passage, M, at the back of the muffle, instead of opening into the furnace, is independently connected with the main flue by a small flue, N (regulated by a suitable damper, P, at its mouth). (No damper has been fitted close enough to effect this.)

4. The increased width of about 4 in. given to the whole furnace is not added to the grate area but is taken up by a single-brick wall Q, which divides the grate into two equal parts and supports the muffle, thus making two separate furnaces with a common flue. By this arrangement the fuel falls evenly on each grate, requiring no trimming beyond the necessary poking to keep the bars free of ash.

5. The flue opening, H, is about 2 in. wide and the same length as the muffle. The sides, GG, are brought down to within an inch or two of the top of the muffle, compelling the hot gases to pass quite evenly over it, front and back being uniformly heated.

This narrow flue enlarges into a chamber R, from the back of which is the off-take to the vertical flue.

This enlargement assists in the desired longitudinal distribution of the draught.

6. The top of the dividing wall, Q, is bevelled so as to support the muffle along the centre without cutting off more than an inch of its width from the direct heat of the fire. This inch width,

however, gets heated by conduction as quickly as the greater part of the muffle and is never liable to sudden cooling.

7. A damper turning on a rod of iron as an axle with a lever and chain attached is preferred to a sliding one, first, because it is easier to move, and secondly, because it affords only a very small area of possible leakage of air alongside the axle into the flue, whereas a sliding damper with only $\frac{1}{16}$ in. clearance gives a considerable leakage. There is also less vibration and shaking with the liability to break the joints in the brickwork.

Two furnaces constructed in this way have been employed for about three years, at Messrs. Henderson & Son's Assay Office, Enys Quay, Truro, and have been found very easy to manage, a very uniform temperature being maintained from end to end, it being possible by means of the independent flue, N, to maintain a sufficient current of air in the muffle when the mouth is as nearly closed as possible. By this means the air becomes heated as it enters and the front of the muffle is not chilled.

The time required to get up the heat at the commencement of the day's work is very much lessened through the absence of clinker, for the same reason coke is saved, and on account of the improved support the life of the muffle is very much lengthened.

The author's thanks are due to Mr. Maitland Kneebone and other pupils of Messrs. Henderson & Son for the painstaking labour they have bestowed on the preparation of the wall diagrams.

DISCUSSION.

Mr. Frank Owen said that he had seen the furnace in question, and quite believed it was all Mr. Martyn claimed. If he had to put up a muffle furnace, however, it would not be that described by the author. He thought that an improved furnace for burning charcoal was of more importance than one for burning coke, especially for those who had to make assays abroad, where charcoal, and bad charcoal at that, had to be employed.

Mr. McDermott thought that the furnace described by the author got over a good many disadvantages. Even with charcoal, clinkers were liable to form, and the muffle might break in out of the way places, which would be a serious difficulty. He considered that the paper was a very valuable one, and impressed upon the members the desirability of sending in short notes relating to any little labour-saving arrangement or device, both as regarded mining and metallurgy.

"Notes on the Occurrence of Mica in South Norway."

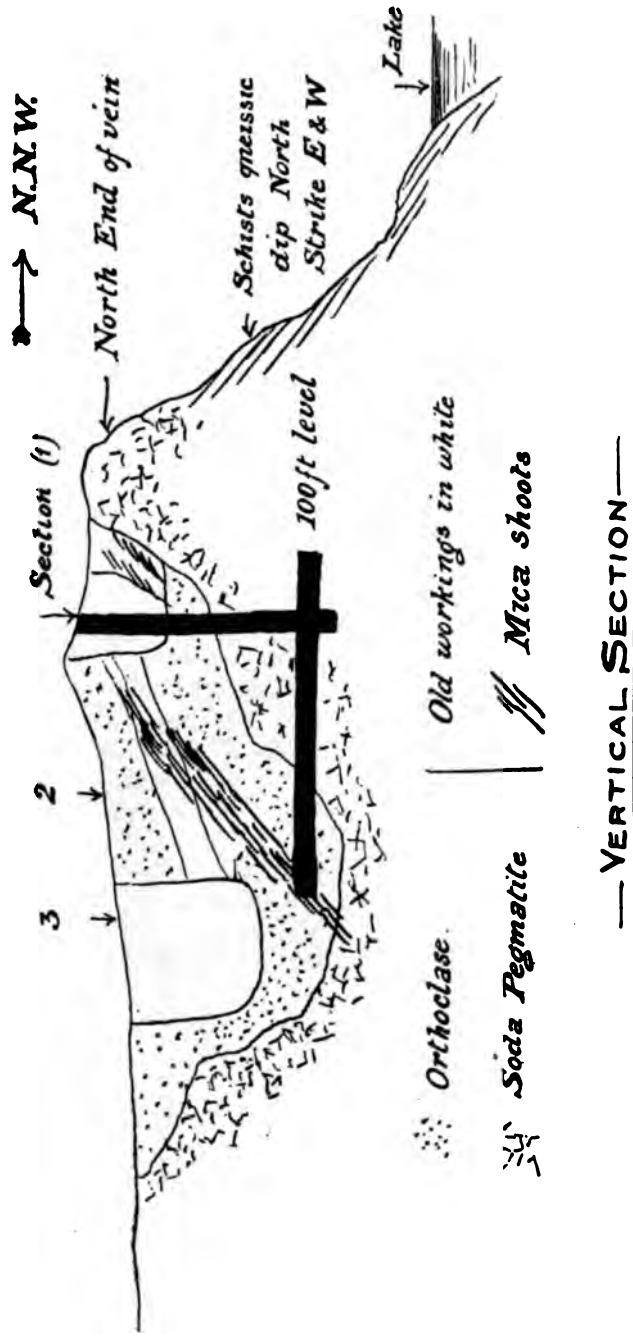
By J. F. Wells.

Godfjeld Mine is situated some 8 miles north-west of Kragerö, on the south-west coast of Norway. It was originally worked for felspar, and subsequently for mica.

The rocks of the surrounding country are mica schists, gneisses, hornblende, and tourmaline schists.

These schists are much broken up by numerous intrusions of eruptive granite dykes of a class usually designated as pegmatite veins. Their general strike is N.N.W. and S.S.E., and they usually dip towards the south, but frequently are nearly vertical. These veins or dykes are of two distinct classes, viz., those which break through or cross the strike of the schists; and, secondly, those which occur as veins or lenses between the foliations of the schists. The mineral constituents of both are similar, and are essentially the same as those that occur in the granite. Both soda and potash felspars are present, and are easily distinguishable. The distribution of the minerals is very uneven. In places masses of orthoclase occur both as masses without definite shape and as huge crystals, and adjoining them is always found large quantities of quartz, but as a whole the veins are composed of pegmatite, with the pegmatite structure very highly developed. The only mineralogical difference between the two types of veins is the occurrence of many rare minerals in the first named, and their absence in the veins lying parallel with the schistose foliations.

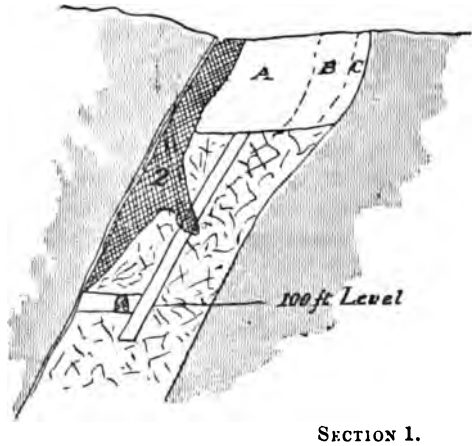
At Godfjeld the vein is a large one, varying from 14 to 27 ft. in thickness, and having the usual strike and an underlay of about 77° with the horizon. The outcrop, however, can only be traced for some 200 yards, as at the south end it is covered over by schists, and at the north end it finishes abruptly some 180 ft. above the lake at the foot of the hill. On the other side of the lake, which is here about 200 yards across, the vein reappears, but its character has altered considerably, when it reaches the top of the hill on this side of the lake it runs into a granite outcrop. The strike of the


N.N.W.
North End of vein
Schists gneissic
dip North
Strike E & W
Lake
Section (1)
100 ft level
2
3
Old workings in white
Mica shoots
Orthoclase
Soda Pegmatite
VERTICAL SECTION

schists in the immediate neighbourhood of the vein is at an angle to the strike of the vein, being approximately W.N.W. and E.S.E., and the dip N.E., but the strike as a whole is very variable. The chief mineral contents of the vein are orthoclase, a soda felspar, quartz, and muscovite. Accessory minerals are apatite, tourmaline, biotite, garnet, columbite, orangite, thorite, &c.

With the sketches of the deposit the writer hopes to be able to make clear the distribution of the mineral contents of the vein, and the position of the mica therein. Fig. 1 represents the section of the north end of the vein along the line of the strike. Three sections across the vein are given, and their positions are marked on this figure.

On the hanging wall side of the vein there was a deposit of milky quartz, of a vitreous and somewhat glassy nature, being non-crystalline; this quartz did not extend to the hanging wall of the vein, as a whole, but was separated from it by a mixture of the main constituents of the vein, both potash and soda felspars being present, chiefly the former. This quartz extended along the strike of the vein for a distance of about 120 ft., and had the same dip as the vein, but dipped also south at about 45° with the horizon. The thickness varied from 2 to 12 ft. On the footwall side of the quartz crystals of orthoclase occurred, subtranslucent, and of a pale brown colour. The crystals were of large size, some measuring as much as 4 ft. across and 5 to 6 ft. high, but this occurrence of felspar was rather erratic. At times it was remarkably free from other minerals, but it often was full of small mica and blebs of amorphous quartz. As the footwall was reached it gradually

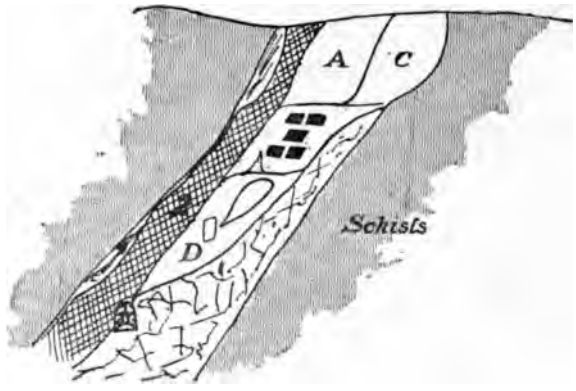


A. Orthoclase—to north of this small shoots of mica.

B. Mixed orthoclase and fine mica, small quartz, and tourmaline.

C. Mixed soda and potash felspars, biotite, muscovite, &c.

SECTION 1.

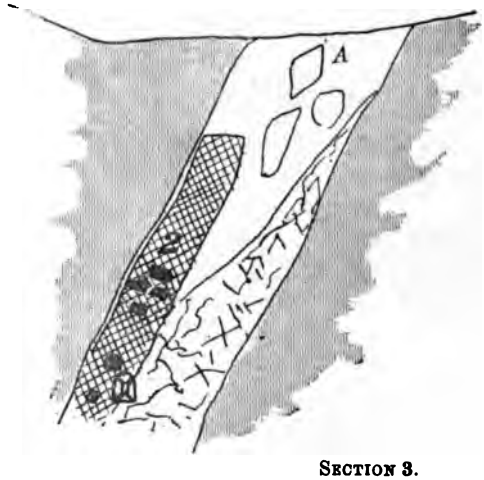


Mica.

D. Crystals and patches. Orthoclase.

2. Quartz.

SECTION 2.



A. Orthoclase, mostly 2nd quality.

2. Mica, rotten, much decomposed.

SECTION 3.

became of worse quality, and finally its place was taken by a soda felspar pegmatite. As shown in the figures, this soda pegmatite gradually absorbed the whole vein; the north end of the vein being entirely composed of this pegmatite which dipped south, and finally made its appearance on the surface at the extreme south end of the outcrop.

There was a very marked distinction between the two felspars, not only from a crystallographic point of view but also in that the two felspars never seemed to merge into each other, there being not only a difference in colour but often the boundary between the two

would be the face of an orthoclase crystal. The cleavage of the soda felspar was not always apparent; often it seemed to be in a semi-amorphous condition, enclosing blebs of non-crystalline quartz.

The mica (muscovite) which formed a lens or shoot, lying between the quartz and the felspar, or wholly in the quartz, occurred as large crystals or as roses of crystals, varying in size from a few pounds in weight up to 400 or 500 lb.

This shoot varied in dimensions. At times it extended almost across the vein, but was, as a whole, either between the felspar and the quartz or entirely in the quartz, and its colour was green; but the greater part of the mica had a brown coloration, due to the numerous stains of rutile enclosures or growths in the crystals. The size of the plates obtainable varied from small pieces up to sheets measuring 2 ft. by 1 ft.

Near the surface the mica was much stained, but it improved in depth down to about 50 ft. Very frequently much disappointment was caused by the outside appearance of the crystals, as, on splitting them it would be found that they were valueless owing to enclosed garnets, tourmaline, and numerous secondary growths of mica. The tourmaline crystals generally were found lying between the cleavage of the plates, yet often they penetrated the crystals parallel with the vertical axis. The secondary growths of mica, though usually crystalline, had no definite position of their axes in regard to those of the older crystals.

The large crystals of mica had a very interesting but at the same time objectionable peculiarity. The two basal planes were never parallel to each other, and, consequently, the majority of plates split out of the mother crystal were thicker at one end than at the other.

At a depth of about 60 ft. the character of the mica began to change very considerably. Although there still were crystals which yielded a small quota of plates, the greater part of them were either rotten or so intergrown that they would not cleave, but broke up into flakes; further, a good many crystals had undergone an entire change, and a greenish, steatitic substance took the place of the mica. This substance gave reactions for alumina, magnesia, potash, and silica. In places it was compact, in others it became earthy, or occasionally fibrous and silky in nature. Shortly after exposure to the air it hardened. This substance did not contain either garnets or tourmaline, though the adjoining mica was full of small garnets.

At the lowest place in the workings, a little below the 100 ft. level, there was still mica, but it was very rotten, and practically of no commercial value.

The mica when first taken from the mine was very soft, but after a few months the mica loses this property to some extent, and becomes much harder and less pliant.

A few particulars of the other minerals occurring in the vein may be of interest.

Apatite in hexagonal crystals occurred here and there: they were often of considerable size, 4 and 5 in. across, and as much as 2 ft. in length. They occurred either in the quartz or in the potash felspar.

Tourmaline occurred in all parts of the vein, but never of a quality good enough to be cut into gems.

Orangite and thorite occurred on the footwall side of the vein in patches, and these patches usually contained parts of all the heavier minerals occurring, tourmaline was always present.

Traces of copper pyrites were occasionally met with, and the schists in the neighbourhood of the mine often carried traces of copper pyrites.

Labour was cheap and of fair quality, wages were as follows:—

		kr.		kr.
Labourers	per 10 hrs. day	2·00	to	2·50
Miners	,, ,, ,,	2·75	,,	3·25
Smiths and carpenters	,, ,, ,,	2·75	,,	3·50
Boys	,, ,, ,,	0·50	,,	1·50
Women	,, ,, ,,	1·00	,,	1·25
Foremen	,, ,, ,,	3·50	,,	4·50

Dynamite costs 2·25 kr. per kilogram.

Boring was usually done by contract work, and the following prices were paid per foot—

In quartz	25 to 30	öre.	
,, felspar	25	,,	
,, proximity of mica.....	30 ,, 40	,,	
,, pegmatite	30 ,, 45	,,	

Working costs varied, including mining, explosives, hauling, sorting, and management, from 2 to 4 kroner per ton of 2,240 lb.

With regard to the value of the various products, &c.,

Felspar, 1st quality, c.i.f. Stettin	26·50	kr. per ton.
,, 2nd quality, ,, ,, ...	14·50	,, ,,

No quotation for mica is given, as the writer considers that any prices given may be misleading, and he finds that no two people agree as to the price of the same sample, but, according to the size

the value may be from a few pence per lb. up to 100*s*. for special sizes.

The mica trade seems to be in the hands of a few, who have their own methods of dealing with any lots placed upon the market. On the other hand, there is no doubt that, by working a retail business from the mine good prices can be obtained, but still the majority of people who dabble in mica seem invariably to burn their fingers.

Note.—£1 equals 18 kroner; 1 kroner equals 100 öre.

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